VOL. XXXV 1919

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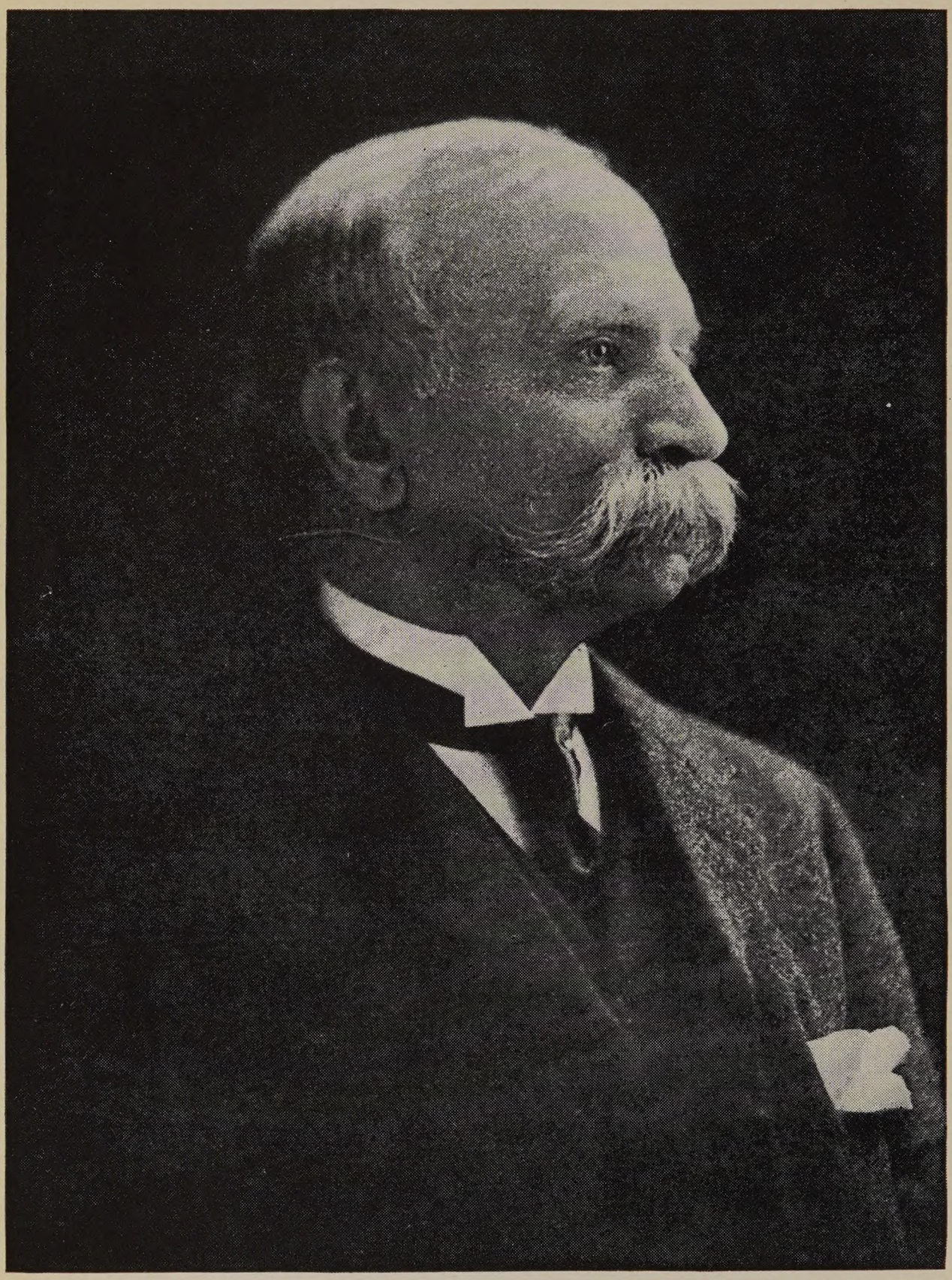

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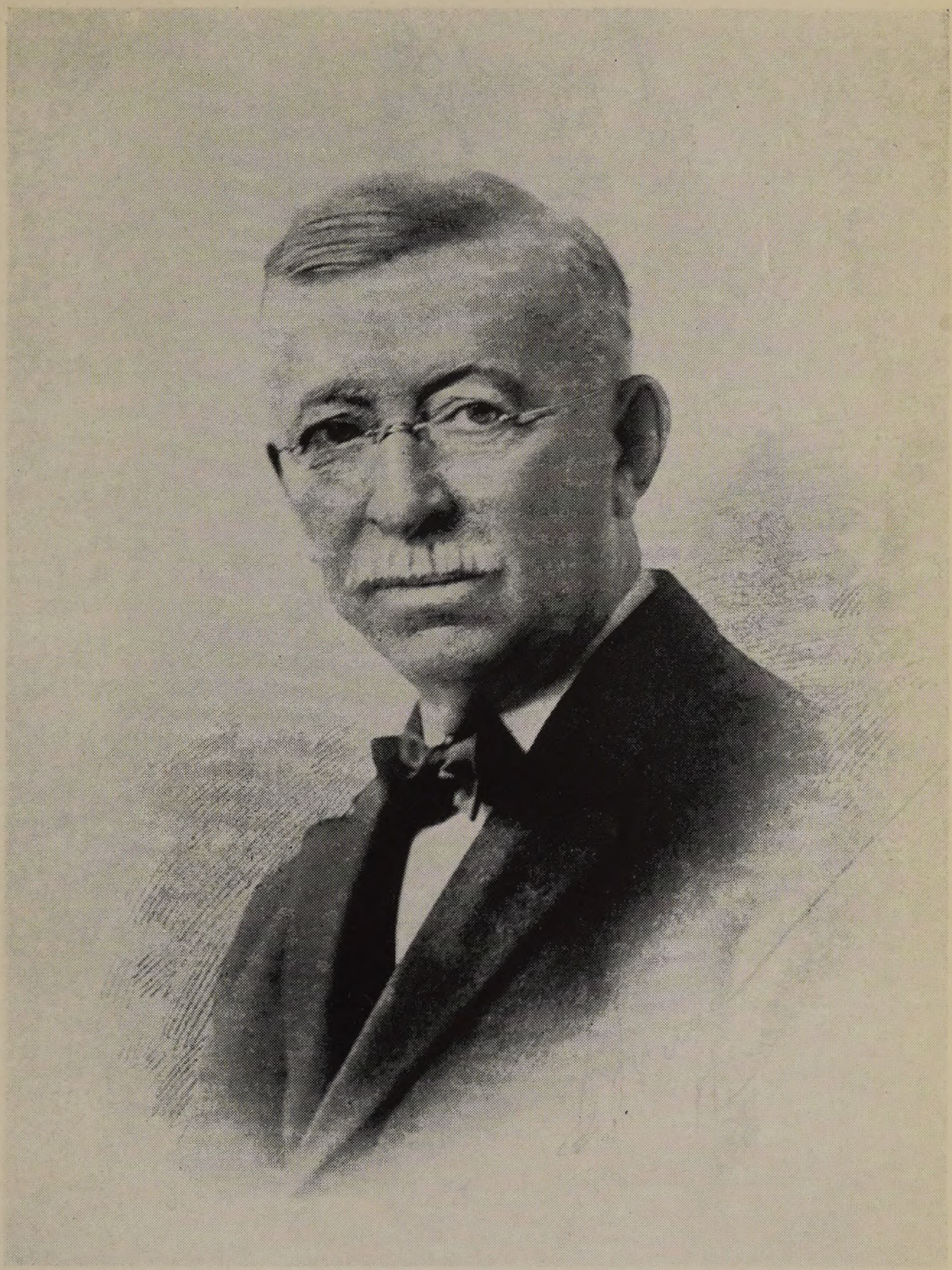

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VOLUME XXXV

THIRTY-FIFTH GENERAL MEETING
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VOLUME XXXV 1919

TRANSACTIONS

OF THE

AMERICAN ELECTROCHEMICAL SOCIETY

PROCEEDINGS

CONDENSED MINUTES OF THE THIRTY-FIFTH GENERAL MEETING OF THE SOCIETY, HELD IN NEW YORK CITY, APRIL 3, 4 AND 5, 1919.

Number of members registered, 213; number of guests registered, 123; total, 336.

PROCEEDINGS OF WEDNESDAY, APRIL 2

While the meeting of the Society did not commence until April 3d, registration was started at 6.00 P. M. Wednesday, April 2d, in the lobby of the Chemists' Club. At 8.30 P. M. the Board of Directors held its regular business meeting.

PROCEEDINGS OF THURSDAY, APRIL 3

President F. J. Tone opened and presided over the annual business meeting of the Society in Rumford Hall, Chemists' Club, at 9.30 A. M. After making some remarks on the progress of the Society during the past year and its present satisfactory condition in the matter of membership and finances, the regular order of business of the annual meeting was taken up.

Secretary Richards read the annual report of the Board of Directors, accompanied by the annual report of the Secretary and the Treasurer. These reports were accepted and are appended to the Proceedings of this meeting.

The Chair announced that at the meeting of the Board of Directors held the previous evening there were elected as honorary

members, Dr. Charles F. Chandler, Professor Emeritus of Chemistry of Columbia University, and Dr. Edgar F. Smith, Provost of the University and Professor of Chemistry of the University of Pennsylvania. The Chair congratulated the Society on heading its honorary list with the names of two such eminent scientists.

Next the President called for the report of the Committee on the Naval Consulting Board, which was read by Secretary Richards. The complete report will be found appended to these Proceedings.

After this the Chair asked for the report of the Committee on Algebraic Signs of Potential. In response, Dr. Hering, the Chairman of this Committee, stated that he had not succeeded in getting the Committee together to decide on the final form of the report and that he thought it best, therefore, to postpone matters until the Committee can have a meeting and send in a final report. Dr. Fink moved that, *pro tem.*, the publications of the Society adopt the negative sign of the zinc electrode, until such time that the matter shall be more definitely settled. The motion was seconded and passed.

The Chair then called for the report of the Committee on New Patent Legislation. In the absence of Dr. Baekeland, the Chairman of our Patent Committee, Mr. E. J. Prindle spoke for him on the subject of the new patent legislation, as proposed by the National Research Council. Concluding, Mr. Prindle offered a resolution that the Society approve the report of the Patent Committee of the National Research Council and the enactment of the legislation proposed therein, and direct that said report be printed in the Transactions of the Society. The motion was duly seconded and unanimously adopted. The report is appended to these proceedings.

President Tone then asked Mr. FitzGerald to present a report of the Public Relations Committee on Government Ownership of Water Power. This report, which had been approved by the Board of Directors, has been made part of the Transactions of the Society, with a paper on the same subject by Mr. FitzGerald appended thereto.

The Chair called on the Secretary to read the resolution adopted by the Public Relations Committee and approved by the Board of Directors, on Water Power Surveys. The resolution follows:

Resolution on Power Resource Surveys.

"WHEREAS, Secretary Lane, of the Interior Department, sent to the last Congress requests for two appropriations for special investigations and reports on power supply—one of these being for an appropriation of fifty thousand dollars for a survey of the power resources of the United States, the work to be done by the Geological Survey, and the other for an appropriation of two hundred thousand dollars for a report on the power supply for the industrial region of the Northern Atlantic Seaboard, extending in general from Boston to Washington, this work to be done by the Geological Survey and the Bureau of Mines in co-operation, and

WHEREAS, The American Electrochemical Society is vitally interested in the full development of water powers and in the proper utilization of all of the energy resources of the United States;

RESOLVED, That this Society approves of the proposed legislation, that it urges that measures covering these surveys be introduced in the next session of Congress, that it authorizes the Committee on Public Relations to support this legislation and assist in its enactment in any way the Committee may find advisable, and that a copy of this Resolution be sent to Secretary Lane and to the Chairman of the House Committee on Appropriations."

The Secretary then presented the report of the Publication Committee, which was approved and is appended to the Proceedings of this meeting.

There being no further reports of committees, the Chair called for the report of the Tellers of election, which was submitted as follows:

REPORT OF TELLERS OF ELECTION

Following is a list of votes cast in the election of officers for the year 1919-1920:

President: W. D. Bancroft, 248; C. G. Schluederberg, 74; W. R. Whitney, 1; F. C. Frary, 1.

Vice-Presidents: J. V. N. Dorr, 266; W. R. Whitney, 236; Carl Hering, 217; A. H. Hooker, 119; W. H. Walker, 116.

Managers: H. C. Parmelee, 212; R. E. Zimmerman, 135; E. Blough, 127; L. H. Baekeland, 115; F. C. Frary, 91; W. H. Walker, 71; A. H. Hooker, 65; F. A. Lidbury, 60; H. B. Coho, 45; J. W. Beckman, 21; W. R. Whitney, 21; F. A. J. FitzGerald, 1.

Treasurer: P. G. Salom, 321.

Secretary: J. W. Richards, 317; R. P. Hommel, 1.

Signed:

BARRY MACNUTT,
ALLISON BUTTS,
S. S. SEYFERT.

The President announced the following elections, as the result of the report of the Tellers:

President: W. D. Bancroft.

Vice-Presidents: J. V. N. Dorr, W. R. Whitney, Carl Hering.

Managers: H. C. Parmelee, R. E. Zimmerman, Earl Blough.

Treasurer: P. G. Salom.

Secretary: J. W. Richards.

The President then introduced the incoming President, Colonel W. D. Bancroft, who took the Chair while the retiring President delivered his presidential address. The address is printed in full as the first paper in these Transactions.

The presidential address delivered, reading and discussing of a paper by E. F. Northrup was taken up, as recorded in full in these Transactions.

After a short adjournment for luncheon papers were presented by the following and discussed, as printed in these Transactions: Gustave Gin, G. G. Griswold, G. A. Guess, O. P. Watts (two papers), H. D. Braley and C. F. Carrier.

In the evening Mr. W. S. Landis lectured in Rumford Hall on the Oxidation of Ammonia to Nitric Acid. This lecture, together with the ensuing discussion, is printed in these Transactions.

PROCEEDINGS OF FRIDAY, APRIL 4

The meeting was called to order at 9.30 A. M. by President Tone, in Rumford Hall. The whole day was devoted to a "Symposium on Released Information," with a short intermission for luncheon. Papers were presented by the following and discussed

as printed in these Transactions, with the exception of the paper by Colonel Walker, which has been printed elsewhere: O. Hutchins, W. H. Walker; G. A. Richter (abstracted by W. D. Bancroft), W. L. Argo, F. C. Mathers, B. Humiston and C. O. Anderson (abstracted by W. D. Bancroft), A. B. Lamb (abstracted by Major Wilson), C. A. Doremus, R. A. Wilson and W. G. Horsch, B. G. Klugh (abstracted by Secretary Richards), A. G. Reeve, T. F. Baily.

After the conclusion of the symposium two more papers were presented by W. E. Moore and G. K. Elliott respectively. These papers, with their discussions, are likewise printed in these Transactions.

Dr. Carl Hering then offered the following resolution of thanks, which was unanimously adopted.

RESOLUTION OF THANKS

Moved that the American Electrochemical Society extends its thanks officially to the Chemists' Club, for the hospitality extended to the Society during this Thirty-fifth General Meeting, to the Loose-Wiles Biscuit Company, the Wright-Martin Aircraft Corporation, and the Nichols Copper Company, for their kind invitations to visit their plants, to the Local Committees of the Society and of the Local New York Section, for their time and effort in so successfully arranging the details of the Meeting, and to Section Q for its prospective entertainment this evening.

The retiring President Tone then, after a few well-chosen remarks, declared the thirty-fifth General Meeting adjourned.

In the evening the members and guests attended a Smoker tendered by the New York Section. The entertainments were under the auspices of Section Q, which by no means fell short of its high reputation for the cleverness and brilliancy of its entertainments.

PROCEEDINGS OF SATURDAY, APRIL 5

The day was spent in visits to the plants of the Loose-Wiles Biscuit Co., the Wright-Martin Aircraft Corporation and the Nichols Copper Co. The participants in these visits were conducted in large sight-seeing motor busses to these plants and expressed their high satisfaction with the interesting trip.

REPORT OF THE REPRESENTATIVE OF THE SOCIETY UPON THE NAVAL CONSULTING BOARD

As the official nominee and representative of the American Electrochemical Society on the Naval Consulting Board of the United States, the undersigned begs to submit the following Report of the Board's activities during the year ending December 31, 1918. It will be realized that on account of the confidential nature of the duties performed by the Naval Consulting Board, its members would not be justified in making a full report as to its activities. We are therefore obliged to confine ourselves to the statements made concerning the Board by the Secretary of the Navy in his last Annual Report:

Anti-Submarine Devices.

"The Department has been keenly appreciative of the necessity of employing every effort to develop devices with which to combat the enemy's submarine operations. Advices and suggestions were freely requested and carefully considered by the department, the Naval Consulting Board, the National Research Council, the various bureaus and departments, the General Board of the Navy, and other boards in Washington.

"In order to co-ordinate the efforts in this matter the Department appointed a special board of officers to have complete charge of carrying out experiments on antisubmarine devices. The General Electric Co., the Western Electric Co., and the Submarine Signal Co. patriotically offered every assistance in this work, and with a fine spirit of co-operation have rendered valuable service. An experimental station was established at New London, Conn., under the supervision of the special board, and many scientists of unusual attainments were assembled there in connection with the experiments and developments of devices for detecting submarines.

"The naval activities at New London, Conn., embrace also a station for assembling and installing various devices developed and training the personnel for the efficient employment of such devices afloat. The results achieved by these methods have been most gratifying; and the operations of vessels fitted with these devices both abroad and in home waters, have had a marked effect in restricting submarine operations."

Developing Inventions and Devices.

"The work of the Naval Consulting Board has increased greatly in both importance and volume during the past years. In addition to work especially adapted to naval uses, the members have assisted in almost every field of research, and Mr. Thomas A. Edison, President of the Board, is still devoting his entire time and his laboratory to war work.

"More than 110,000 ideas, inventions and devices have been considered by this Board and the Department, either separately or conjointly; there have been 4,200 interviews on inventions and upwards of 700 models have been examined. In addition, at the Chicago branch office 2,300 interviews have been granted and 2,500 communications investigated. This gives an idea of the volume of this work and the care that is taken in considering and investigating every invention or suggestion that gives promise of being of any value."

Bulletin on Submarine and Aviation.

"A very complete bulletin on the enemy submarine has been prepared, after the most careful study and compilation of all data, and was issued under date of May 1, 1918. This gave all but the most confidential information on the subject and has served as a guide to the country at large in making suggestions. A second bulletin on the subjects of aeronautics, composed of articles prepared by a number of the best authorities on the subject, was issued on August 1, 1918. This pamphlet included a carefully prepared bibliography that is complete and exhaustive and gives the student in aviation a most valuable fund of information. During the past year the War Committee of Technical Societies, composed of representatives of the great engineering societies, associated themselves with the Naval Consulting Board, and their chairman, Mr. David W. Brunton, was elected a member of the Board. With the erection and occupancy of the new Navy Building space was provided for the resident member and a staff of examiners directly connected with the office of the Secretary of the Navy, and all the files and correspondence have been concentrated in one location, bringing the examination and consideration of ideas and inventions in closer touch with the Navy Department.

"The office of the Board is in charge of Mr. Brunton, who is also a member of the advisory committee to the inventions section of the General Staff of the Army, and there is established a close working basis of handling all ideas presented to the attention of the proper authorities with the least effort and by a very direct route. This has not only resulted in increased efficiency, but in a reduction of expenditure of funds and time consumed."

Subjects That Have Been Studied

"Among the subjects studied by the Board were: Aeronautics, propellers for airplanes, stabilizing and controlling devices, and an instruction machine that has been adopted for use; optical glass, improvements in quality and production; ordnance equipment; anti-submarine defensive devices and offensive instruments; internal combustion engines, refined hydrocarbon fuel for airplane engines giving greater efficiency and longer life to machines; and methods of coal combustion looking to higher economy.

"When the Department called for classes of electrical and mechanical engineers from which to select a number of commissioned officers for active service, the Board was of material assistance in its prompt and efficient handling of the situation, and as a result the Navy was enabled to obtain an excellent selection of able and experienced officers.

"The officers and members of this Board give their time and talents, many of them to the exclusion of their private business, from a desire to render a patriotic and disinterested service. They are entitled not only to the thanks of the Department, but of the large number of other agencies and individuals that have benefited by their counsel and advice."

Very truly yours,

LAWRENCE ADDICKS.

REPORT OF THE PATENT COMMITTEE TO THE NATIONAL RESEARCH COUNCIL

The Commissioner of Patents in 1917, with the approval of the Secretary of the Interior, requested the National Research Council to appoint a committee to investigate the Patent Office and patent system, with a view to increasing their effectiveness, and

to consider what might be done to make the Patent Office more of a national institution and more vitally useful to the industrial life of the country.

Mr. Thomas Ewing, who is a member of your Patent Committee, was the Commissioner of Patents who took that action.

The National Research Council, complying with the request, appointed a Patent Committee, consisting of: Dr. William F. Durand, Chairman; Drs. L. H. Baekeland and M. I. Pupin, scientists and inventors; Drs. R. A. Millikan and S. W. Stratton, scientists; Dr. Reid Hunt, physician; and Messrs. Frederick P. Fish, Thomas Ewing, and Edwin J. Prindle, patent lawyers. On the departure of Dr. Durand for Europe, Dr. Baekeland was appointed acting chairman of the Committee.

Your Committee has approached its work in the belief that the American patent system has been one of the most potent factors in the development of the prosperity of our country. Americans, being descendants of the European races, are not naturally more inventive than are Europeans, but under the incentives of the American patent system they have produced many more inventions and been able to pay high wages and live on a better scale than Europeans.

American inventions have played a vital part in the war. There is hardly any implement or explosive that our Army and Navy has used which is not more or less the result of American invention. The Patent Office is keeping secret and withholding for publication many inventions made since the beginning of the war and which are useful in war. After the war, it will be imperative that American inventors continuously improve American products and the manufacture of them, and make basically new inventions to meet and keep ahead of the strenuous efforts which Germany and other nations will make to attain supremacy by these methods.

Your Committee has, therefore, carefully investigated the Patent Office and the patent system, with a view to increasing their effectiveness, and, based on its investigation and the experience of its members, makes the following recommendations:

The Committee has concluded to propose a program consisting of but four features, because it believes those features are of such fundamental importance that their enactment into law would strengthen the entire system and directly and indirectly establish

it upon a new and much more advantageous footing before Congress and the public; and because with a simple program, presenting comparatively little opportunity for difference of opinion as to the desirability of the changes proposed, there would be a unanimity of opinion in support of it which could not be obtained if the program were more extended.

A Single Court of Patent Appeals.

The first proposal which your Committee recommends is the establishment of a single Court of Patent Appeals that will have jurisdiction of appeals in patent cases from all the United States District Courts throughout the country, in place of the nine independent Circuit Courts of Appeals in which appellate jurisdiction is now vested.

Until 1891 the Supreme Court of the United States was the appellate court in patent cases for all the lower courts. At that time the right of appeal to the Supreme Court in patent cases was taken away, and that court now hears patent cases only upon writs of *certiorari,* which are never granted unless certain very unusual conditions exist.

The existence of nine appellate courts of concurrent jurisdiction in patent cases works serious hardships. While, theoretically, the law is the same in all of these courts, there has been an irresistible tendency to drift apart in the application of the law. It has even happened in a substantial number of cases that two of the appellate courts have taken a different view of one and the same patent. It is, of course, very important that the questions which always exist as to the validity and scope of a patent should be settled once and for all at the earliest possible date in the life of the patent, for, as a practical matter, seventeen years (the term of a patent) is a comparatively short time in which to reduce the invention to a thoroughly commercial form, to prepare for its manufacture, and to introduce it upon the market, and it is usually necessary to determine the validity and scope of the patent in order to determine the amount of money which it is safe to invest in exploiting the invention. As things are now, whichever party succeeds in the first suit that is tried on the patent, the other party is very likely to feel that in a second trial before another court he might have better luck. He, therefore, is inclined to

insist upon a second litigation. Meantime, he advertises that the questions involved were not settled in the first case. This means uncertainty on the part of the owners of the patent as to their rights and uncertainty on the part of the public as to its rights to use the invention or to determine what it must avoid in working in the same field—a really intolerable situation.

Moreover, we shall never have a uniform and definite patent law, consistently applied, until we have a single Court of Patent Appeals independent of local sentiment, realizing a responsibility to fix the principles of the law and enforcing an harmonious application of these principles on the lower courts. It would be of the utmost value to those in the United States who are engaged in industry if the present confused condition could be corrected and a single tribunal devote itself to crystallizing the fundamentals of the patent law and to educating the courts throughout the land to uniformity in applying these principles in special cases.

Attached hereto is a copy of a bill for the establishment of such a court, which has been advocated for many years by the American Bar Association, and is No. 5011, of the House of Representatives, 65th Congress, 1st Session. It provides for a court of seven members, which would sit in Washington, with a Chief Justice appointed for life by the President. The appointment of the Chief Justice for life is in order that there may be an element of continuity in the court. The other judges are to be selected by the Chief Justice of the United States Supreme Court from the various district and circuit judges throughout the land, and each is to sit on the Court of Patent Appeals for a period of six years, or longer, if reappointed.

There are many advantages in this plan. Among them are the following:

The judges would not be men who were appointed as judges primarily to deal with patent matters. There could be no charge that special interests had a hand in their selection or that they were chosen to promote special views as to the patent law and its application. They would be men who had been primarily selected by the President as fit to be federal judges in the localities where they live. Federal judges are men of a high type, and many of them are broad-minded men, much respected in the communities where they serve. They would take up the work of

the Court of Patent Appeals with a breadth coming from the performance of their general duties of judges in their own circuits or districts and would, therefore, escape the narrowing which so often comes from continuous work in a specialized field.

The Chief Justice of the United States Supreme Court would select from the district and circuit judges throughout the land men whom he thought most competent to serve for a term on the Court of Patent Appeals. We would seldom, if ever, take more than one judge at a time from any one circuit. The Court, therefore, would be made up of men who were primarily judges and who would be recognized as bringing to the Court of Patent Appeals the instincts and feelings, on the subject of the interpretation of the patent law, of the courts and of the people in the communities in which they live.

Undoubtedly many of them would be on the appellate court for one term only and after that they would go back to their circuits or district with a training as patent judges such as could be obtained only by sitting for a period of years in such an appellate court. They would not only be qualified as patent judges, but they would reflect the atmosphere of the appellate court and cause the atmosphere to pervade their own neighborhood. They would thereafter undoubtedly be selected to hear patent cases in the lower courts in preference to judges who had not had training in the Court of Patent Appeals. The courts throughout the country would, in time, become educated to the high and definite standards established by the Court of Patent Appeals, not only by study of the decisions of that court, but by the presence in the lower courts of men who had had this special training in the upper court.

It is of the utmost importance that these judges in the Court of Patent Appeals should be well paid. Otherwise they might not be willing to break up their homes and go to Washington for a limited term. We think that their salaries should be higher than those of the judges of any court in the United States except the United States Supreme Court.

The increased expense due to such a court would be small. The aggregate amount of work to be done by the judges of the United States courts as a whole would not be changed to any substantial extent, because all appeals must now be heard by the present courts and judges, and, if there were a single Court of Patent

Appeals, the Court of Appeal in the nine circuits would be relieved of just as many appeals as were heard by it. The judges in some of the circuits are much overworked, but this is not true of many of the circuits. The Chief Justice of the United States Supreme Court, in selecting these judges, could, if he chose, take into account the work of the different circuits and whether one circuit or another could best spare a judge.

As the law now stands, judges from one circuit may be called upon, and not infrequently are called upon, to go into other circuits which are short-handed. In this way, any undue pressure upon the judges in any particular circuit, by reason of the loss of any single judge who went to the Court of Patent Appeals for six years, could be relieved.

Moreover, it is no hardship to increase the number of judges where necessary. The whole judicial system of the United States is said not to cost as much as it does to run one first-class battleship, and the addition of a few judges would be a negligible burden upon the Treasury.

A further advantage of a single Court of Patent Appeals would be that it would see clearly where there were defects in the statute and in the conditions and practice in the Patent Office, and could speak with authority on all matters which affect the theory and practical working of the patent system.

The Patent Office a Separate Institution and Independent of the Department of the Interior.

The second proposal which your Committee recommends is that the Patent Office be made a separate institution, independent of the Interior or any other department.

The Patent Office was originally in the State Department, but, on the formation of the Interior Department in 1849, is was made a bureau of that department and has been so ever since.

The only matters connected with the Patent Office with which the Secretary of the Interior has anything to do are the following: The Secretary of the Interior must submit to Congress all estimates for appropriations. All appointments, excepting those of the Commissioner, two Assistant Commissioners, and five Examiners-in-Chief, are made by the Secretary, but only on the recommendation of the Commissioner. The eight places named are

presidential appointments, but the Secretary makes recommendations to the President. All matters of disbarment or reinstatement after disbarment of attorneys are passed upon finally by the Secretary. All matters of discipline are under the Secretary's jurisdiction. The Secretary of the Interior must approve all changes in the Rules of Practice of the Patent Office, but he cannot compel the Commissioner to make any change whatsoever.

No appeal lies to the Secretary from any decisions of the Commissioner, either in matters of merit or practice. All such matters, as far as they are reviewable, rest with the courts of the District of Columbia.

The Secretary of the Interior no longer signs the patents, and has no jurisdiction to grant or refuse them.

Thus, it will be seen that the Secretary of the Interior is not required to know anything about patents or patent law. He is not selected because of any qualification for the granting of patents or supervision over the Patent Office. The Secretary of the Interior has less influence over the Patent Office than over any other bureau of the Interior Department, because there are appeals to him from all the other bureaus. Nor is the Patent Office related to any other bureau of the Interior Department.

The Secretary of the Interior has recently moved out of the Patent Office building, thus severing physical contact with the Patent Office, which is but a type of the lack of mental contact between the office of the Secretary of the Interior and the Patent Office.

The experience of many commissioners over a period of several generations has shown that, no matter how pleasant the personal relations may be, the Commissioner of Patents cannot expect any real benefit to the Patent Office to flow from its connection with the Interior Department. There is nothing in common between the interests of the Interior Department and those of the Patent Office, and, consequently, nothing to produce any advantage from the amalgamation of the Patent Office with the Interior Department.

Your Committee believes that to make the Patent Office an independent bureau would greatly increase the respect of the

public and Congress and the courts for it, and would make it easier to procure enlarged appropriations and better salaries than under present conditions.

As to the appropriations, under present conditions the demands of the Patent Office for equipment, personnel, and salaries are necessarily subjected to comparison both by the Secretary of the Interior and by Congress with those of several other unrelated bureaus, each pressing its own demands and criticizing any apparent preference. In the opinion of your Committee, this operates as a severe handicap. In estimating the needs of the Patent Office there should be no discussion of the demands, for example, of the Pension Office or the General Land Office. As an independent institution, the needs of the Patent Office would be judged on their necessity and the appropriations be determined by consideration of general policy.

As to personnel: the enhanced dignity and independence of the Patent Office would render all positions of importance in it more attractive, and particularly would make it easier to secure and retain in office men of the necessary qualifications to fill the difficult office of Commissioner.

A copy of a proposed bill for making the Patent Office an independent bureau is annexed to this report and its enactment is recommended by your Committee.

Increase in Force and Salaries of the Patent Office.

The third proposal which your Committee recommends is a substantial increase in the force and salaries of the Patent Office. The patents granted by the United States Patent Office are of less average probable validity than formerly, because the number of applications for patent and the field of search are constantly increasing, while the examining force for many years has been insufficiently large and has not been increased proportionately. The inducements are so unattractive that 25 percent of the examining force has resigned within the past three years. Your Committee finds that the Patent Office is suffering both from lack of examiners and from inadequate compensation.

The salaries of the Patent Office examiners have been increased only ten percent since they were fixed in 1848, when they were approximately the same as those of members of Congress. At

the time the salaries of the Examiners-in-Chief were fixed, they were the same as those of Federal District Judges. During the past seventy years, the compensation for technical service in almost all other directions has been increased very largely. Congress, in creating new positions, is willing to pay technical men salaries more nearly approximating the usual compensation of such men in private service, but, having started a position at a given salary, is very loath to increase the salary. A Principal Examiner, to pass the entrance examination for the Patent Office, must himself have an education equivalent to that of a college graduate, and yet his salary is so low ($2,700 a year) that it is practically impossible for him to give his own sons a college education.

Your Committee believes that salaries should be paid to the examiners proportionate to those paid for equally high technical work in other departments created recently; such, for example, as are paid in the Army and Navy and in the office of the Attorney General. The examiners are passing upon questions often involving millions of dollars, and they cannot be at their best in this vitally important work unless their salaries are large enough for them to live comfortably and without strain. The chances of making mistakes in the granting of patents are great enough even under the most favorable circumstances, and they should not be increased by compelling the examiners to work for inadequate salaries. The inducements should be such as to present compensation and a career which would attract and hold men of the highest ability. The payment of adequate salaries and the creation of provisions tending to hold out attractive prospects to the examiners would also tend to raise the dignity of the Patent Office and to increase its standing in the estimation of the public and of Congress and the courts, and so would tend to enhance the value to the public of the patent system.

The work of the Patent Office has grown so much more rapidly than has the examining force that the examination to determine whether or not the invention claimed in an application for patent is novel is imperatively restricted to the field of search where it is most likely that the invention will be found. Many patents are granted which would not be granted if the examiner had time to make a thorough search. One of the Assistant Commissioners

of Patents is compelled to devote a large amount of his time to speeding the work of the examiners in order to prevent further falling behind in the number of unexamined cases. Money is often invested on the strength of patents, only to find later that the patent is upset in the courts, because the Patent Office search did not go far enough to discover that the invention had already been disclosed in some earlier patent or publication. The granting of a patent with invalid claims or claims which are too broad or which are nebulous is a menace to the art to which it relates, and until such a patent has been adjudicated and its effect judicially determined, it tends to prevent manufacturing and commerce in that art. Such a patent may, in this way, cost the public many millions of dollars beside the cost of establishing its invalidity or its true breadth or meaning by litigation, and the prevention of the granting of such patents by any reasonable increase in the examining force of the Patent Office would, in many cases, be a very large saving. The inducement to inventors and investors in patents is consequently lessened, the standing of patents before the courts and the public is impaired, and the production of inventions discouraged.

Your Committee accordingly recommends a substantial increase in the salaries of the Patent Office officials, and in the number and salaries of the Examiners, as provided in the draft of a bill for that purpose which is attached hereto.

While your Committee believes the Patent Office so fully justifies its existence that it would be an exceedingly profitable investment, even though all expenses were paid from the public income, the Patent Office has always been self-supporting and the increase in salaries and examining force which the Committee recommends can easily be entirely taken care of by the Patent Office income, if necessary.

Compensation for Infringement of Patents.

While an injunction can ordinarily be obtained against an infringer in a case where a patent is adjudged valid, except where it would interfere with government work, a money recovery had not heretofore been generally possible except under most favorable circumstances. In a case where it cannot be said that the entire salability of the article depends upon the invention, it has

been necessary to show just how much of the price of the article is attributable to the invention, and as it is ordinarily impossible to make such a separation, and as most patent cases are ones in which it cannot be said that the whole salability of the article depended upon the invention, it has resulted that recovery of money is seldom obtained in a patent suit.

Recently there have been two or three decisions in which the courts have taken a more liberal attitude, holding in effect that where an invention has been used by an infringer a reasonable royalty may be awarded to the patentee based on a mere estimation or on opinion evidence, even though no exact computation can be made. This is analogous to the attitude of the courts in personal injury cases and is entirely just and reasonable. While, as stated, there have been two or three decisions to this effect, it may take a generation to induce United States courts generally to adopt this position, if at all, and the Committee therefore proposes that the law be amended to provide, that as damages to the complainant, the court, on due proceedings had, may adjudge and decree to the owner payment of a reasonable royalty or other form of general damages. Such an amendment has been provided in the attached bill amending Section 4921, the Revised Statutes of the United States, and reading as follows:

If proof is not offered, or, in the absence of adequate proof of the amount that should be awarded as damages or profits, the court, on due proceedings had, may adjudge and decree to the owner payment of a reasonable royalty or other form of general damages.

This proposed amendment would enable the patentee in all suits where the patent has been found valid and infringed to recover at least a reasonable royalty, and would provide a money recovery in the great majority of patent suits where no recovery would otherwise be possible. The Committee believes that the comparative certainty of financial return would answer one of the most common and strongest reproaches against the patent system, namely, that a patent does not ordinarily pay the inventor any money, and it believes that the incentive to invent would accordingly be greatly increased.

There are some cases in which it seems to many who are familiar with such matters as though the courts were inclined to

go to the other extreme and award damages out of all proportion. Where a complaint has shown that profits have been made by the use of an article patented as an entirety, the infringer is liable for all the profits unless he can show—and the burden of proof is on him to show—that a portion of them is a result of some other invention used by him. If the infringer cannot show what proportion of the profits is due to such other invention, then all his profits must go to the complainant. Any rule by which the entire profits are given to a patentee in the absence of proof that they are all due to the invention of the patent sued upon, is unfortunate and sometimes very unjust. The proposed amendment to the statute would permit a court under these circumstances to do substantial justice even though it could not be mathematically exact. In other words, the amendment to the statute would enable a court to avoid awarding either too much or too little.

Conclusion.

Your Committee, believing that the American patent system is vitally useful in our system of Government, therefore recommends that the reforms herein discussed be enacted into law.

Your Committee also recommends that this report be approved by the National Research Council and that the Committee be continued for the purpose of arousing and co-ordinating interest in and support for the necessary legislation of various national societies, manufacturing interests, bar associations, and other elements of the public.

L. H. Baekeland, *Acting Chairman,*
William F. Durand, *Chairman,*
(absent in France)
M. I. Pupin,
R. A. Millikan,
S. W. Stratton,
(See reservation below)
Reid Hunt,
Frederick P. Fish,
(See reservation below)
Thomas Ewing,
Edwin J. Prindle.

Approved: James T. Newton, Commissioner of Patents.

Reservation by Dr. Stratton.

I agree to the terms of the report with the exception of that portion which refers to the establishment of the Patent Office as a separate government institution. It is not quite clear in my own mind that this would be the best thing to do since in general it is best for all government establishments to be represented in the Cabinet.

(Signed) S. W. STRATTON.

Reservation by Mr. Fish.

I entirely concur in the substance of the conclusions set out in the above report.

I think, however, that the words "if proof is not offered, or" in that portion of proposed Section 4921 which deals with damages and profits, should be omitted so that the sentence in which those words appear should read:

"In the absence of adequate proof of the amount that should be awarded as damages or profits, the Court, on due proceedings had, may adjudge and decree to the owner payment of a reasonable royalty or other form of general damages."

I do not think that a statute should directly or indirectly contemplate a condition in litigation in which "proof is not offered." I believe that the clause which I suggest would accomplish the desired purpose and that the courts in applying the clause would be embarrassed if the phrase "if proof is not offered" were in the statute.

I think also that general damages by way of a reasonable royalty or otherwise should not be awarded unless it appeared that actual damages or actual profits due to the unlawful use of the invention could not be determined, and that there should not be any language in the statute which implied that no effort be made to determine such actual damages and profits.

(Signed) FREDERICK P. FISH.

Reservation by Mr. Newton.

Regarding the report of the Patent Committee, after careful consideration I have concluded it best to withdraw my approval of that part of the report concerning the separation of the Patent Office from the Interior Department.

I hope we will all exert ourselves for the passage of these statutes in proportion to the importance of the subject.

(Signed) J. T. NEWTON.

REPORT OF THE PUBLICATION COMMITTEE

During the year 1918-1919 there were thirty-one papers submitted to this Committee for consideration. These have been disposed of as follows:

Accepted twenty-five, accepted after modification two, rejected two, under consideration two. Total thirty-one.

Of the two rejected papers an appeal was made to the Board of Directors in the case of one of them and the Board overruled the Committee's action on the ground that the author had been to considerable trouble and expense in preparing the paper.

FRANCIS A. J. FITZGERALD, *Chairman.*

ANNUAL REPORT OF THE BOARD OF DIRECTORS

To the Members of the American Electrochemical Society:

The following are some of the important items of business transacted by your Board of Directors during the past year:

It was ordered that the names of members of the Society who are in military service be retained on the rolls without payment of dues until the close of the war, at which time such members may again receive the Publications of the Society upon recommencing payment of dues; meanwhile the Bulletin was ordered sent to such members.

One hundred dollars was appropriated for the expense of preparing an exhibit for the Society for the Fourth National Exposition of Chemical Industries, the same to be placed in the hands of Mr. C. G. Schluederberg for that purpose.

The Treasurer was authorized to subscribe $2,000 to the Third Liberty Loan from cash on hand.

The Treasurer was authorized to subscribe $2,000 to the Fourth Liberty Loan from cash on hand.

The Board accepted with a hearty vote of thanks a contribution of $325.00 from the Finance Committee of the Thirty-second General Meeting, held at Pittsburgh, Pa., being the surplus of local contributions above expenses of holding the meeting.

One thousand dollars was appropriated during 1918 for the purposes of the Membership Committee to further the campaign for new members.

The Society contributed to the expenses of the War Committee of Technical Societies fifteen cents per member, making the contribution for 1918 $247.50.

This contribution was later returned intact upon the disbandment of the War Committee of Technical Societies.

The Board accepted as a basis of exchange with the Faraday Society that we will furnish members of their Society with our Transactions at $2.25 per annum for paper covered copies; the Faraday Society conversely agrees to furnish members of this Society, desiring them, with their Transactions at the same rate —$2.25 per annum.

The following resolution was passed by the Board: *"Resolved,* That it is the sense of this Board that all members of the Society who are enemy aliens and who are in sympathy with the enemy of the United States of America in the present war, should be dismissed from membership."

An annual contribution of $50 was made to the publishing of the Annual Tables of Physical and Chemical Constants.

SECRETARY'S ANNUAL REPORT

To the Board of Directors of the American Electrochemical Society:

GENTLEMEN: In 1918 the Society held two general meetings, one on a trip through the Appalachian South, April 28th to May 5th, at which the attendance was 107 members and 28 guests, total 135; the second at Atlantic City, N. J., September 30th to October 2d, at which the attendance was 82 members and 32 guests, total 114. The Transactions of the spring meeting include 24 papers, and those of the fall meeting, 20 papers.

So far Volume XXXIII has been distributed, containing 318 pages. Volume XXXIV will be ready for distribution in about two months.

The edition of Volume XXXIII was 1,400 copies bound in cloth for distribution to our members; 50 extra copies for distribution

in pamphlet form to authors of papers; 250 copies unbound to be kept in stock; 250 copies bound in paper for distribution to the Faraday Society.

The stock of Volumes on hand December 31, 1918, was as follows:

Volume	Bound in Cloth	Bound in Paper	Total
I	131	13	144
II	136	0	136
III	87	0	87
IV	77	233	310
V	38	234	272
VI	177	10	127
VII	62	195	257
VIII	55	321	376
IX	80	315	395
X	85	244	329
XI	52	266	319
XII	62	254	316
XIII	149	0	149
XIV	146	17	163
XV	191	223	414
XVI	190	288	478
XVII	97	519	616
XVIII	107	522	629
XIX	186	331	517
XX	184	334	518
XXI	184	354	538
XXII	183	304	487
XXIII	215	293	508
XXIV	333	0	333
XXV	284	238	522
XXVI	4	547	551
XXVII	29	538	567
XXVIII	6	551	557
XXIX	158	555	713
XXX	236	28	264
XXXI	160	0	160
XXXII	172	233	405
Index	286	68	354

Financial Statement.

Receipts in 1918.

Cash Assets beginning 1918 account		$2,633 86
Entrance fees	$1,615 00	
Current dues	7,320 00	
Volumes, 1918	4,390 00	
Back dues, 1917	720 00	
Volumes, 1917	325 00	
Advance dues, 1919	815 00	
Volumes, 1919	455 00	
Advance dues, 1920	10 00	
Sale of publications	1,666 65	
Sale of reprints	137 50	
Interest on bank account	128 30	
Interest on Liberty Bonds	224 80	
Interest on Philadelphia Electric Bonds	155 00	
Sale of Society pins	5 00	
Gift of Pittsburgh Section	325 00	
Subscription for Faraday Society	324 00	
Surplus on Southern Trip	141 97	
		18,758 22
		$21,392 08

Expenditures in 1918.

Secretarial appropriation	$2,100 00	
Office printing	876 10	
Printing of Volume 32	2,898 24	
Printing of advance copies and discussions	1,637 54	
Engraving	392 28	
Extra reprints	122 69	
Binding of volumes	376 93	
Office postage	665 06	
Postage on advance copies and volumes	704 00	
Office expenses	454 66	
Expenses of meeting	810 12	
Office equipment	123 65	
Storage and insurance	237 25	
Membership Committee	590 25	
Publication Committee	27 05	
Booth Committee	151 46	
Local Sections	425 00	
Freight, insurance and duty on Faraday Society Transactions	155 20	
Contribution to War Committee of Technical Societies	247 50	
Annual Tables of Physical and Chemical Constants	50 00	
Miscellaneous	10 00	
Investment in Liberty Bonds	4,000 00	
		$17,054 98
Cash assets beginning January, 1919, account		4,337 10
		$21,392 08

Condition of Membership of the Society in 1918.

Members January 1, 1918		1,680
Elected and qualified as members in 1918		345
		2,025
Resignations in 1918	31	
Deaths in 1918	15	
Dropped for non-payment of dues	76	
		122
Members December 31, 1918		1,903

Condition, April 2, 1919.

Members January 1, 1919		1,903
Qualified as members to April 2, 1919		99
		2,002
Deaths	1	
Dropped for non-payment of dues	73	
		74
Members April 2, 1919		1,928

TREASURER'S ANNUAL REPORT, 1918

January 1, 1918, cash balance	$2,633 86	
Total receipts for 1918	18,758 22	
		$21,392 08
Total expenditures, 1918		17,054 98
Balance, December 31, 1918		$4,337 10

We have examined the above statement of account of receipts and expenditures for the year 1918 and find the same to be correct.

We also report the following securities of the Society in the Safe Deposit Vault of the Commonwealth Title Insurance and Trust Co., Philadelphia, Pa.:

Philadelphia Electric Co. 5's	$3,100 00
1st Liberty Loan	2,000 00
2d Liberty Loan	2,000 00
3d Liberty Loan	2,000 00
4th Liberty Loan	2,000 00
	$11,100 00

CARL HERING,
S. S. SADTLER,
Auditors.

MEMBERS AND GUESTS REGISTERED AT THE THIRTY-FIFTH GENERAL MEETING

Members.

E. G. Acheson
Quinton Adams
Geo. P. Adamson
Lawrence Addicks
D. K. Bachofner
T. F. Baily
A. T. Baldwin
W. D. Bancroft
F. E. Bartell
Chas. Baskerville
Wm. H. Beck
E. H. Bedell
H. E. Behnken
E. O. Benjamin
Edw. L. Blossom
Wm. Blum
Linn Bradley
G. F. Brindley
M. J. Brown
N. B. Brown
N. W. Buch
C. F. Burgess
Allison Butts
Geo. Campbell
J. N. Carothers
C. F. Carrier, Jr.
S. C. Carrier
Fred. E. Carter
S. N. Castle
R. T. Chace
J. B. Chaffe, Jr.
N. K. Chaney
H. C. Chapin
R. B. Chillas
W. R. Clymer
C. F. Coffin
H. B. Coho
W. P. Cohoe
E. F. Cone
H. C. Cooper
F. T. Cope
Wm. A. Cowan
G. E. Cox
F. D. Crane
Ed. L. Crosby
F. W. Cunningham
N. E. Dabolt
R. W. Davis, Jr.
D. J. Demorest
E. S. Dewey
Jos. L. Dixon
C. A. Doremus
J. V. N. Dorr
H. H. Dow
Saul Dushman
Geo. K. Elliott
Carleton Ellis
Harrington Emerson
H. B. Faber
W. B. Featherstone
C. G. Fink
F. A. J. FitzGerald
R. Fleming
J. A. Fogarty
H. W. Fox
J. J. Frank
F. C. Frary
Raymond Freas
W. H. Fulweiler
D. P. Gaillard
R. H. Gaines
J. J. Gerety
Wm. H. Gesell
F. H. Getman
A. E. Gibbs
C. B. Gibson
A. McK. Gifford
H. W. Gillett
G. C. Given
H. M. Goodwin
J. S. Gravely
J. H. Gray
Wm. H. Grosvenor
L. O. Hart
J. Hedalen
Carl Hering
Chas. H. Herty
Harold Hibbert
A. T. Hinckley
Alcan Hirsch
G. B. Hogaboom
R. P. Hommel
W. G. Horsch
Henry Howard
L. E. Howard
Burr Humiston
Frederick Hurum
O. Hutchins
W. McA. Johnson
Lyman C. Judson
F. T. Kaelin
Thomas J. Keenan
Edw. Keller
H. W. Kellogg
Philo Kemery
E. F. Kern
D. B. Keyes
Rolf Knudsen
C. N. Laird
C. A. Laise
W. S. Landis
P. E. Landolt
S. C. Langdon
H. R. Lee
F. A. Lidbury
F. G. Liljenroth
A. D. Little
E. A. Lof
A. Lodyguine
E. G. Love
W. R. Loveman
J. R. MacMillan
Duncan Mac Rae
C. P. Madsen
C. G. Maier
W. D. Mainwaring
C. W. Marsh
A. B Marvin
E. J. K. Mason
J. A. Mathews
J. Y. McConnell
W. A. McKnight
J. D. McManus
H. S. McQuaid
M. W. Merrill
D. D. Miller
Chas. Mindeleff
Richard Moldenke
W. E. Moore
E. E. Mueser
I. Murai
J. M. Muir
E. F. Northrup
H. C. Parmelee
Chas. L. Parsons
N. A. V. Paulsson
N. Petinot
O. W. Pickering
J. B. Pierce, Jr.
E. C. Pitman
E. J. Prindle
F. R. Pyne
A. G. Reeve
H. T. Reeve
Jos. W. Richards
C. N. Richardson

Members (Continued.)

L. T. Richardson
E. G. Rippel
F. W. Robinson
C. F. Roth
G. A. Roush
D. B. Rushmore
C. J. Russell
H. D. Ruhm
F. J. Ryan
L. E. Saunders
C. G. Schluederberg
J. E. Schott
B. H. Schubert
F. F. Schuetz
E. Kilburn Scott
J. A. Seede
H. J. Skinner
S. Skowronski
Acheson Smith
Dyer Smith
Buckner Speed
H. N. Spicer
H. M. St. John
Wm. H. Stone
F. A. Strauss
F. G. Sefing
D. P. Smith
E. M. Symmes
Kozo Tabata
Y. Tada
C. E. Taylor
H. S. Taylor
J. E. Teeple
L. S. Thurston
F. J. Tone
S. A. Tucker
R. Turnbull
W. S. Tyler
M. Unger
G. D. Van Arsdale
E. C. Walker
Wm. H. Walker
A. T. Ward
J. T. Ward
E. Weintraub
F. S. Weiser
J. H. White
R. H. White
Brent Wiley
E. M. Wilkins
H. H. Willard
C. A. Winder
W. J. Winninghoff
W. W. Winship
M. Yano
J. L. Yardley
Carl Yunck
J. A. Yunck

Guests.

E. G. Acheson, Jr., Niagara Falls, N. Y.
E. G. Anderson, New Britain, Conn.
G. K. Anderson, Philadelphia, Pa.
H. O. Andrew, New York City
R. H. Applegate, New York City
J. Asahara, Tokyo, Japan
Guy Atkinson, New York City
W. E. B. Baker, New Haven, Pa.
H. E. Bakken, Pittsburgh, Pa.
Chas. E. Bateson, New York City
H. A. Bedworth, Waterbury, Conn.
Mrs. E. O. Benjamin, Newark, N. J.
D. D. Berolzheimer, New York City
Geo. J. Blum, New York City
C. H. Booth, Chicago, Ill.
Frederic Bounet, Jr., Landing, N. J.
C. S. Brainin, Newark, N. J.
W. K. Brown, New Haven, Conn.
O. Byron, New York City
G. H. Buchanan, Elizabeth, N. J.
P. H. Butler, New York City
Mrs. J. N. Carothers, Washington, D. C.
John H. Carson, New York City
Mrs. L. W. Chaney, New York City
Mrs. N. K. Chaney, Cleveland, Ohio
Mrs. R. H. Chaney, New York City
H. W. Charlton, New York City
H. O. Chute, New York City
A. B. Conner, Detroit, Mich.
S. K. Dahl, Venice, Calif.
H. V. Day, Jersey City, N. J.
H A. Doerner, Denver, Colo.
Miss Antoinette N. Dorr
H. F. Durham, New York City
M. C. Eilman, Montreal, Canada
Chas. H. Eldridge, Yonkers, N. Y.
Mrs. Harrington Emerson, New York City
Miss M. E. H. Emerson, New York City
P. B. Findley, New York City
Mrs. Colin G. Fink, Yonkers, N. Y.
C. M. FitzGerald, New York City
I. W. Gross, New York City
A. M. Hageman, Bloomfield, N. J.
J. H. Harris, New York City
M. Hauber, Jr., West Haverstraw, N. Y.
Ellwood Hendricks, New York City
C. W. Hill, Pittsburgh, Pa.
Miss Grace Hillabrand, Chicago, Ill.
A. K. Hobby, Boston, Mass.
G. A. Hodge, Sao Paulo, Brazil
Frank Hodsen, Philadelphia, Pa.
G. B. Hogaboom, Jr., New Britain, Conn.
Wm. O. Hutchins, Keene, N. H.
W. Hutton, London, England
W. A. Janssen, Montreal, Canada
Morres Joffe, New York City
John Johnston, New Haven, Conn.
G. H. Jump, Buffalo, N. Y.
A. W. Keller, New York City
Stewart Kerr, Darby, Pa.
K. A. Kilbourne, Elizabeth, N. J.
T. Kimishima, Tokyo, Japan

Guests (Continued)

Z. Kimura, New York City
K. G. Knox, New York City
Th. Krempovsky, Russia
Dr. W. A. Lef, Philadelphia, Pa.
S. M. Ling, New York City
H. V. Lipps, New York City
E. W. Lloyd, Chicago, Ill.
E. E. Lyder, New York City
James Mann, New York City
C. R. Marchant, Baltimore, Md.
Mrs.C.W. Marsh, Greenwich, Conn.
Harry P. Martin, Niagara Falls, N. Y.
E. P. McCrorhen, New York City
James D. Miller, Harrisburg, Pa.
Reay Miles,
A. C. Morrison, New York City
Mrs. E. E. Mueser, New York City
Jacques S. Negru, New York City
Arthur C. Neish, New York City
A. W. Nickerson, New York City
S. Nishikawa, Tokyo, Japan
Mrs. E. F. Northrup, Princeton, N. J.
A. B. Oatman, Niagara Falls, N. Y.
J. C. Olsen, Brooklyn, N. Y.
F. F. Patero, New York City
H. D. Pease, New York City
H. S. Pickens, Philadelphia, Pa.
Harold B. Pickering, Arlington, N. J.
E. S. Porter, Arlington, N. J.
H. W. Porth, Jersey City, N. J.
L. N. Reed, Philadelphia, Pa.
G. H. Reid, New York City
V. V. Rieser, Jersey City, N. J.
F. D. S. Robertson, Toronto, Can.
W. B. Roscovich, New York City
H. Rosenthal, New York City
Mrs. G. A. Roush, Bethlehem, Pa.
W. J. Russell, New York City
H. Satoh, Tokyo, Japan
W. H. Schleisinger, Denver, Colo.
E. E. Schumacher, New York City
Fr. Sejersted, Kristiania, Norway
W. R. Seyfried, Elizabeth, N. J.
R. M. Shaw, New York City
R. N. Shreve, New York City
T. F. Slattery, New Britain, Conn.
A. E. Soisson, Kenvil, N. J.
W. H. Steinberg, Passaic, N. J.
Waldemar M. Sternberg, Rolla, Mo.
Mrs. H. M. St. John, Chicago, Ill.
H. F. Tenney, Rome, N. Y.
A. E. Thurber, New York City
M. R. Timmer, Hackettstown, N. J.
S. L. Tyler, New York City
L. Unger, New York City
W. G. Waitt, Cleveland, Ohio
E. M. Watts, New York City
A. E. Wells, Washington, D. C.
R. E. Wilson, Boston, Mass.
J. H. Wright, New York City
Geo. J. Young, New York City

MEMBERS ELECTED AND QUALIFIED

Since Issuing Last Membership List (January 31, 1919).

February 19, 1919:
F. C. Atkinson
Shigekata Hori
Alexander Silverman
Jose Balta de Cela
P. T. Vanderwaart
Chuji Yabe
Dyer B. Lake

February 24, 1919:
Alex. DeMilan
A. B. Andrews
I. B. Simonsen
M. W. Wilkinson
V. V. Kelsey
A. L. Muren
A. W. Demmler
Chas. H. Wood

March 20, 1919:
Montford Morrison
G. A. Grille, Jr.
H. G. A. Tesch
N. B. Tan
L. P. Ohliger
Kisaburo Yamaguchi
Thos. F. Judge
Fusanori Ohashi
S. C. Langdon
Hirota Satoh
H. L. Wallau
R. U. Fitting
A. E. Case
F. T. Kaelin
Harrison Hale
W. R. Barrett
L. H. Conklin

March 20, 1919: (Cont.)
E. L. Hauman

March 24, 1919:
Chas. S. Wagner
J. S. O'Donovan
C. T. Evans
Jules Cousin
J. H. Reedy
A. H. Warth
C. E. Speirs
P. B. Williamson
G. L. Peakes
H. M. Armstrong
R. R. Cole
J. C. B. Baldwin
Frank S. Smith
D. C. Burroughs

Members Elected and Qualified (Continued).

March 24, 1919. (Cont.)
A. C. Hewitt
I. W. Edwards
R. M. A. Lloyd
J. M. Drabelle
H. W. Brooks
W. R. Adair
B. F. Kohlhausen
H. E. Wheeler
W. H. Graef
G. S. Davis
F. C. Clark
F. E. Lee
J. B. Douthit
F. N. Bosson
E. S. Hight
C. J. Macke
C. Q. Wiggins
A. T. Hyde
D. C. Roumbanis
F. D. S. Robertson
J. B. Hutchison
F. B. Steele
H. H. Beatty
F. P. Vaughan
Antonio Rins Miro
Arthur D. Stephanus
F. W. Cohen

March 31, 1919:
J. M. Scott-Maxwell
F. B. Flinn
J. W. Lewis
H. S. Wilkins
C. M. Thompson
M. C. Gilman
Hideo Kai
W. C. Cram, Jr.
L. N. Reed
D. W. Blakeslee
E. E. Smith
Jos. C. Clark

April 17, 1919:
C. A. Bray
T. E. Hendrick
F. A. Fair
M. F. Massey
G. H. Reid
E. L. Huff
Chas. H. Eldridge
L. E. Westman
G. R. Swany
J. B. Chaffee, Jr.
N. H. Gellert
W. F. Edwards
C. H. Howell

April 17, 1919: (Cont.)
D. W. Hoot
C. H. Eshbaugh
R. A. Uihlein
G. J. Lipscomb
M. T. Kummer
R. W. Mitchell
L. W. Ganschow
Geo. Campbell
E. P. Oswald
H. P. Hill
W. S. Lee
Frank E. Ford
A. B. Gerber
L. H. Milligan
A. S. Dana
J. A. Welker
E. J. Mommo
W. M. Sternberg
W. F. Washburn
W. A. Janssen
E. W. Lloyd
H. E. Bakken
Rene de Sallier
John H. Graybill
J. D. Whitmore
D. C. Reed
A. E. Fitz
H. D. Blake
F. P. Matthews
F. W. Durkee
H. M. Kuhn
G. H. Finks
N. M. Hopkins
Walter Dalton
Helge Rost
P. A. McTerney
E. M. Honan

May 8, 1919:
E. F. Collins
T. Eardley-Wilmot
C. W. Hill
O. A. Havill
Geo. T. Southgate
W. S. Burch
T. J. Cecere
E. J. Simon
W. S. Norton
G. D. Muggleton
J. N. Junkins

May 12, 1919:
A. R. Meyer
M. J. Bottinelli
O. H. Eschholz
Paul Poetschke

May 12, 1919: (Cont.)
F. H. Miller
W. R. Woolley
L. A. Hazeltine
John Johnston
W. S. Roberts
Harold Boericke
Mrs. Mary U. von Isakovics
J. J. Elbert
L. J. Barton
Wm. R. Alley
R. A. Folsom
Mark Eldridge
C. C. Lucas, Jr.

May 21, 1919:
Wm. McKeown
K. R. Brown
N. H. Hunt
Wm. F. Keyes
Julius C. Peterson
J. A. Wales
N. L. Morgan
E. H. Wesle
B. R. Grier
F. R. Knight
R. R. Knowles
C. E. W. Bateson
W. C. Cole
C. B. Willmore
W. W. Lackie
H. V. Churchill
J. R. Spurrier
M. H. Logan

June 25, 1919:
W. J. Livingston, Jr.
C. R. Corey
W. S. Moody
G. Deakin
J. H. Andersen
R. M. Harrington
E. S. Bardwell
H. A. Bedworth
W. G. Soley
A. K. Hobby
E. L. Stenger
J. M. Smith
C. E. Gilman
R. T. Whitnall
Arthur Johnson
O. F. Flumerfelt
T. J. Mateer
C. F. Markness
Irving C. Bull
Oscar T. Coffelt

Members Elected and Qualified (Continued).

June 25, 1919: (Cont.)
A. C. Morrison
H. E. Morse
W. A. Barrows
F. P. Dengler
C. B. Rogers
M. J. Udy
H. A. Beale, Jr.
Wm. H. Hampton
F. R. Wadleigh
W. G. Waitt
F. M. Slough

July 23, 1919:
W. B. Clemmitt
Chas. Robbins
F. H. Mountford
Lee Boyer
George C. Westby
S. T. Leo
C. E. Singer
C. S. Palmer
F. W. Slaten
Chas. Mantel
Carl H. Ruh
F. B. Bayless
J. Härdén
R. G. Hinman
M. F. Peters
E. B. Latham
Rowland King
Will Baughman
R. W. Bissell
L. J. Luckenbach
Chas. R. Kuzell
A. L. Ferguson
H. D. Carr
G. J. Brown
Stephen P. Gould
B. M. O'Harra
Chas. C. Thomas
W. Kobayashi
R. G. Walker
Joseph Slepian
R. H. Kienle
Yoshio Kawakita
J. W. McKim
Hugo Reinbold
C. A. Wright
J. O. Elton
Edw. P. Fleming
James P. Gill
A. D. Buzby
H. C. Arnold
S. F. Shaw
Frank Hodson
Frank B. Downes

August 21, 1919:
Ichiro Ito
K. Takakuwa
R. B. May
Julius Kahn
C. T. Bragg
C. G. Dennis
F. E. Beasley
S. W. Miller
Paul McAllister
Lee Fraser
W. W. Taylor
Crosby Field
Blamey Stevens
F. L. Firebaugh
L. G. Gomez
Joseph Ceader
K. V. Laird
H. A. Winne
F. S. Schimerka
Herbert Martin
James T. Duce
B. Berg-Hansen
Giacomo Merizzi
R. M. Harrington
Z. Kimura

September 18, 1919:
Herbert Speight
Chas. L. Reese
A. H. W. Aten
A. F. Wahlberg
W. Palmaer
Pierce Barker
L. R. Wright
F. W. Clements
Roger W. Ryan
E. W. Rice, Jr.
J. H. Critchett
M. E. Altmeyer
James Colquhoun
B. C. Lamble
S. T. Crookes
A. D. Gardiol
Julius H. Gillis
Henning Broms
Johan Murer
R. H. Gaither
G. W. Prince
W. S. Hendrixson

October 24, 1919:
W.T.Kendall Brown
F. H. Lamb
Y. Shigematsu
Huibert Doyer
A. M. Hageman

October 24, 1919: (Cont.)
W. D. Harkins
Z. T. K. Woo
L. J. Rosenshine
V.K.Srinivasayengar
J. S. Lawrenson
B. Paravicini
L. P. Lee
H. E. R. Philpott
James C. Alley
Guy K. Mitchell
A. D. Bell
W. A. Hiddleson
L. Burpee
J. S. M. Wharton
James B. Woodyatt
Walter C. Pearce
Wm. A. Wong
A. G. P. Palen
A. J. Byington
Mark S. Wilder
J. E. Wright
H. H. Holding
Gadient Engi
C. Y. Wang
J. C. Young
Herbert Winkel
S. Yunoki
T. W. Mather
P. LeG. Reymond
Ivor Kenway
W. E. Harkness
J. G. Butters
John Campbell

November 10, 1919:
Ray E. Tower
Arvid Frisak
W. R. Sammons

November 11, 1919:
L. M. Klauber
Wm. E. Harrison
James C. Munn
Ernest G. Heinrich
R. H. Brownlee
Alvin J. Forbess
Geo. P. McMullen

November 20, 1919:
Wm. V. FitzGerald
Harry Etchells
E. F. Corson
M. R. Bump
W. M. Gifford
E. Caudrelier
Clifton O. Stewart

The Presidential Address delivered at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919.

ELECTROCHEMISTRY IN ITS HUMAN RELATIONS

By F. J. Tone.[1]

During recent years it has been a custom which has now more or less become a dictum that the annual address of the president shall treat of some of the broader aspects of electrochemistry. We have in turn had discourses on the aesthetics of electrochemistry, its external activities, its economic forces and its public relations. I need offer no apology therefore in starting out this morning to make some observations on electrochemistry in its human relations. We are human beings as well as electrochemists, and we may well study the relations of our science to the world problems of today and how we can give most immediate and most effective service for the betterment of human society.

For four years we have been in the thraldom of a great war. We have been passing through a period of violent evolution, the nature and extent of which we have hardly begun to measure or realize. Science and industry and in fact every human activity emerges from this world turmoil with new and rapidly shifting viewpoints and new responsibilities, and electrochemistry forms no exception. It is therefore profitable at this time to take some survey of our present situation and to forecast our future activities and responsibilities in the new alignment of human relationships. If a new era has begun, what is our part in it?

I shall not try to outline any "fourteen points" of future social progress or to say whether some of our values by which we measure such progress are true or false, but one or two things can be stated with a fair degree of assurance. As one of the big results of the war new and better conditions of living are demanded by the masses of the people in all countries. If a higher plane of living in the matter of food, clothing, housing, fuel and sanitation is to be established, and if shorter hours of labor and

[1] President, Carborundum Co., Niagara Falls, N. Y.

more opportunity for recreation and self-improvement are to be the order of the new day, this can only be brought about by vastly increasing the production of all the essentials which enter into the needs of modern society. Not forgetting such factors as co-operation, efficiency of organization and distribution and elimination of wastes, production can be increased principally in two ways: first, by using labor more efficiently, and second, by further supplementing and supplanting labor by the chemical and mechanical instrumentalities of industry. Either we must get more out of labor or we must stimulate the discoveries that will make labor more productive than before. If the world is going to work less, science must work more. The work of the world can not be done in a six-hour day and a thirty-two-hour week unless applied science gets in a lot of time-and-a-half on weekdays and double-time on Sundays.

Among the problems of first importance is that of the world's food supply. The crop yields of the world must be vastly increased, and this in the face of a decreasing labor supply. If cereals, cotton and animal fodder must be produced in greater quantity and more cheaply, the answer can only be found in improved transportation and traction for the farmer and in a vastly extended use of fertilizer, and here rests a big problem with the electrochemist. Judged by European standards the American farmer is inefficient. He produces fourteen bushels of wheat to the acre while in Europe the same acre yields thirty. European yields all along the line average from 50 to 100 percent greater than American yields, and while this increase is in part due to intensive farming, better selection of seed, and crop rotation, the big difference is in the enormously greater use of fertilizer. It is of the highest economic importance that the American farmer shall use more fertilizer.

The three main elements of plant food are nitrogen, potash, and phosphoric acid, and each one of these carries a problem for the electrochemist. The fixation of nitrogen is one of the great triumphs of electrochemistry, and the possible exhaustion of the natural nitrate fields is to us no longer a matter of any anxiety. So long as we have energy and air the land need never want for ammonia. Still the efficiency of the arc process of fixing nitrogen is today less than five percent, and between this figure and the

best efficiency attainable, there is certainly breathing space for the most energetic of our research workers. The extension of the cyanamid process has been one of the remarkable chemical events of the war. Its opportunity for peace-time accomplishment is no less, but we still have some distance to go before artificial nitrates can be laid down to the farmer in such quantity and at such price as will mean the doubling of our crop yield.

The potash problem during the war has been brought into the field of electrochemistry. The Cottrell system of electrostatic precipitation has been successfully applied to gases from blast furnaces and cement plants, and if the waste gases from all such plants were treated by the Cottrell process, the possible recovery of potash would be equal to our pre-war consumption. We can hardly hope that this is commercially possible, but in many localities it will give a permanent source of potash which will maintain its existence under competitive conditions. What the electric furnace can do in phosphoric acid recovery is not so well proved. But passing over this, we see that the solution of our national food problem is one in which, despite our past accomplishments, we still have large responsibilities.

In the new order of things we are going to banish the slums and have more healthful living conditions, and the rôle of the electrochemist in sanitation is told in the story of chlorine. We learn that when the armistice was signed we were just ready to deliver each day to our fighting forces on the western front over two hundred tons of poison gas. The basis of this great offensive weapon was chlorine, and it was denied its opportunity to demonstrate in a big way what it could do to win the war against the Huns, but I believe it has a far greater opportunity in the germ warfare of peace times.

In starting to investigate the work that chlorine is doing in sanitation, I knew in a general way that it was of growing importance in the purification of drinking water, but upon assembling all the data it was astounding to find that chlorine is today making safe the drinking water of more than 20,000,000 people of this country who dwell in cities. Chlorine is safeguarding one-fifth of our population from typhoid. We learn that a military force or an industrial expedition can go anywhere in the tropics and be at all times assured of potable water if it carries a few pounds

of hypochlorite. In the Carrel-Dakin solution for the treatment of wounds, the use of chlorine is defined by a recent authority as the greatest discovery of surgery in the whole war. During the recent epidemic of influenza, the number of cases in the cell rooms of two electrolytic chlorine plants, located in different States, was one-half of that in any other portion of the plants. Starting with a few such facts as these, what more is needed to build up a great structure of organized sanitation with chlorine as its basis. We are looking for an outlet today for a huge producing capacity brought about by the demands of the war. Do not these examples of what chlorine is doing in germ warfare clearly point the way?

The sterilization of sewage is as logical a measure of sanitation as the sterilization of drinking water. To be sure no city drinks its own sewage, but some other city generally does. Why permit Chicago and Milwaukee, Detroit and Cleveland to pollute the waters of Lake Erie, Lake Ontario and the St. Lawrence? Why should Pittsburgh pass its germ-laden water on to Cincinnati and Louisville when an adequate remedy is available at reasonable cost? One hundred and twenty pounds of hypochlorite will sterilize 1,000,000 gallons of screened sewage. A city of 1,000,000 inhabitants can sterilize its sewage at a cost for its chemicals of $360 per day. The cost per single inhabitant for one year would be about twelve cents. But the work of chlorine does not stop with water supply and sewage. It should be used to flush and sprinkle our streets, to clean our cars and public buildings, and to sterilize our dairies. The organized use of chlorine in sanitary milk production will do more to give our children a pure milk supply than any possible system of inspection.

We are searching for an outlet for our excess chlorine in metallurgy and in organic chemicals. If our electrolytic chlorine friends would begin telling people how they can live in a prophylactic environment, how they can feed their babies pure milk, and how they can avoid poisoning their neighbors who live lower down on the same watershed, conditions in the chlorine market would soon be such that they would no longer worry about what the Government is going to do with Edgewood.

Again, in the metallurgy of steel, if electrochemistry has made great contributions to the art, its responsibilities are no less clearly

defined. No one now doubts but that we shall soon attain to the super-steel, and it is no less clearly indicated that this will be accomplished by the electric steel furnace and electrically produced alloys. The triplex process using the electric furnace as the third step for the refining operation produces a steel of crucible quality at a cost of a few dollars a ton more than open-hearth steel. When it is understood that such steel produces rails which do not break and plates which do not fracture, it will require no advertising campaign in the popular weeklies to bring this product into universal use: public opinion will demand it. The alloy steels have made possible the modern automobile, the aeroplane engine and the farm-tractor motor. Alloy steels have been a luxury. They have been used only in the vital parts of the mechanism where enormous strains must be met with a minimum weight of material, but their use must be extended. High quality is not incompatible with tonnage. It is the work of the electrochemist to make possible their use as common materials of engineering, thus contributing vastly to improvements in transportation and the mechanical arts. No one needs to have pointed out what this signifies as an economic gain in modern life.

But the field of alloy steels has by no means been fully exploited. The results which the steel maker has achieved with tungsten, molybdenum, chromium and vanadium, marvelous as they are, by no means mark the limit of the art. We now seem to be on the eve of still more important advances based on the use of new alloys, including ferro-uranium and ferro-zirconium. We hear whispers about a zirconium steel of 300,000 pounds tensile strength and 30 percent elongation. Not being a steel maker and having no metallurgical reputation to lose, I may venture in my imagination to build an ocean liner or a freight train of this super-steel in which the dead weight is cut in half and the carrying capacity is increased 25 percent; or, may I visualize the super-steel bridge and finally the super-steel Pullman car, but in the latter case we are not so anxious to reduce the weight of the car and save steel. Rather we will make the super-steel Pullman four times as strong with the same weight of steel and then sleep soundly in the rear section of the rear car while the second section of the train follows one short block behind. Briefly stated, the

job of the electrometallurgist is to put super-steel on a big tonnage basis.

Let us now consider our energy resources. These are fuel and water power, and their proper utilization will not only reduce the sum total of human toil necessary to run our industrial machine, but will be a measure of vital importance to coming generations. Our energy resources are necessary to the functioning of every other resource. They are at the very basis of industrialism, transportation and domestic well-being. Thus they concern society broadly, but they peculiarly concern the electrochemist because cheap electric power is a basic element in electrochemical production. The electrochemist was responsible for the success of the world's first great hydro-electric development, at Niagara, and became the natural champion of the utilization of our water-power resources. But why have we failed to sell this proposition to the man in the street and to those whom in his wisdom he has chosen to legislate for him. There may be many reasons for public indifference to coal economics and the waste of water power, but the fact remains that we have as yet failed to state the simple facts and principles of our energy economics in a way which carries conviction to the people with whom responsibility for action lies. Here are some of them:

1. Coal is our greatest energy resource. It is irreplaceable, and, notwithstanding an apparent popular belief to the contrary, it is not inexhaustible. The amount available for present and future generations is known to be very definitely limited. The curve of consumption is rising regularly and rapidly, and we can compute with a fair degree of accuracy when our magnificent coal reserves will be exhausted.

2. Coal is wastefully mined, wastefully transported and wastefully utilized. If the bituminous coal used for domestic heating alone was first converted into artificial anthracite and the by-products recovered, it would represent a saving of more than twice our entire output of gold for 1917. If our railroads were electrified and fed from central stations recovering the by-products from the coal, this saving would be doubled.

3. Fuel and water power are practically interchangeable in industrial economics. The failure to use water power is just as truly a waste as would be the burning of our coal mines and oil

wells. If Congress were to enact legislation to set the torch to one-third of all of our coal that was mined in the country last year, it would be no greater act of folly or crime than to obstruct water-power development. The utilization of water-power resources will save coal and oil for future generations and release for other useful work in the present generation hundreds of thousands of miners, vast transportation facilities, and thousands of railway workers.

Having convinced the people that water power must be developed, the most difficult part of the problem remains to be solved, which is to arrive at a form of legislation in which the people and the capitalist believe that their respective interests are protected. Passing over the question of government ownership of water powers, I believe we must all now agree that no legislation for private development stands any chance of enactment which does not carry with it a very much larger measure of public control than would have been acceptable a few years or even a few months ago. Political ideas have changed, and one can plainly observe that public sentiment will not again consent to the development of its water powers as an unrestricted private monopoly. It would not so consent even when awakened to the great economic crime of letting them run to waste. After all, what we want is the power, and I believe this Society can not do less than endorse legislation in which public interests are fully protected by control of rates and proper limitation of charter rights. Following this we want government surveys covering our complete energy resources and their utilization.

My conclusion shall be brief. It is plain that our science faces some of the big human problems of the day. Our opportunity for service was never greater, and I believe it will not be passed by.

A report made by the Committee on Public Relations to the Board of Directors of the American Electrochemical Society and submitted by it to the Society at the Thirty-fifth General Meeting in New York City, April 3, 1919, where it was discussed and adopted by the Society.

GOVERNMENT OWNERSHIP OF WATER POWERS AND ELECTROCHEMICAL INDUSTRY.

It is understood that this report does not consider the question of the government ownership of *sources* of hydro-electric energy nor of government control and regulation of hydro-electric *systems;* but simply operation by the government of hydro-electric systems in comparison with operation thereof by private companies. As regards this aspect of Government Ownership it is the opinion of your Committee:

(1) That anything which affects, either for good or evil, electrochemical industries, will affect in a corresponding way the people as a whole.

(2) That the inherent defects of governmental activities in industrial enterprises will be found in government ownership of hydro-electrical systems and will affect the electrochemical industries dependent on such systems adversely.

(3) That the American Electrochemical Society, being interested in and representing the electrochemical industries of the United States, should strongly oppose all attempts at governmental operation of hydro-electric systems.

It is needless to present any evidence to the members of the American Electrochemical Society that the electrochemical industries are of vast importance to the whole nation and to every individual composing it.

Even a superficial examination of numerous governmental activities shows that those which can only be undertaken by the government leave much to be desired in their results and that when these are of an industrial nature they are invariably inefficient, ineconomical, and subject to numerous glaring defects which are obviously inherent in governmental work of this kind in a democracy, when the present state of human character is con-

sidered. Under an autocratic government certain causes of the defects of such governmental activities do not exist; but as recent experience has shown, even when the existence of a democratic nation is threatened by external aggression and a temporary reversion to autocratic government becomes necessary, governmental industrial enterprise is far inferior to private undertakings of this nature.

It is the belief of all American citizens that governments are instituted among men to secure certain unalienable rights among which not the least is individual liberty and that the reason why their country has prospered so wonderfully is because those intrusted with its administration have not only confined themselves within their constitutional spheres, but have refrained from encroaching on and consequently diminishing the fields of activity of its citizens. Most of our citizens who have given the matter careful thought also believe that governmental encroachment in such fields will greatly overbalance, in permanent evil, any partial or transient benefit which it may yield.

There is no evidence that governmental ownership of hydro-electric systems will be any freer from defects than other kinds of governmental industrial enterprise, but quite the contrary.

Your Committee believes that the American Electrochemical Society should have an opportunity to go on record as opposed to government ownership and operation of hydro-electric systems and that this record should be given the widest publicity.

Signed by

F. J. Tone,	W. H. Walker,
F. A. J. FitzGerald,	Henry S. Carhart,
L. Addicks,	W. R. Whitney,
F. A. Lidbury,	C. G. Fink,
J. W. Richards,	C. F. Burgess.

Opposed to the report, and not signing:

Carl Hering.

An appendix to the preceding Report, presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919.

GOVERNMENT OWNERSHIP OF WATER POWERS AND ELECTROCHEMICAL INDUSTRY.

By F. A. J. FitzGerald.*

The examination of government ownership of hydro-electric developments in relation to electrochemical industry should cover a study of the effect of government ownership on the hydro-electric system considered as a productive industry in itself; the effect of this ownership on electrochemical industries which depend for their supply of electricity on the hydro-electric system; the effect of this ownership on the small user of electricity such as the householder and the nation at large.

Now, as regards the last consideration it is not necessary to do more here then emphasize the fact that electrochemistry has become of such vast importance to the well-being of all the people that anything which affects its development adversely is bound to react on the nation as a whole. This was well brought out in a resolution of the Board of Directors recommended to and adopted by the American Electrochemical Society at its Thirty-first General Meeting in Detroit, May 3, 1917[1]. This report also emphasized the self-evident point that for the successful development of electrochemical industry an abundant supply of cheap electric energy is necessary. Therefore, in this report we may confine ourselves simply to a consideration of the effect of government ownership of hydro-electric systems considered as industrial undertakings.

Arguments in favor of government ownership of hydro-electric systems are based usually on theoretical grounds, because, with an exception to be noted later, we have no good example of this specific kind of government ownership from which to draw conclusions. It is pointed out that the chief item in the cost of

* Electric Furnace Expert, Niagara Falls, N. Y.

[1] Transactions Am. Electrochem. Soc. (1917), 31, 16, *et seq.*

water-power developments is the interest on the investment and that governments can accordingly construct and operate such systems much more cheaply than can private companies. For the government can borrow money at a much lower rate than private individuals can, moreover, since it is not in the business for the sake of profit it will establish suitable sinking funds to pay off all the investment and eventually everyone—the householder, the factory needing mechanical power, the railways, electrochemical industries—will get electric energy at an actual cost which is far below anything which we can now conceive. Furthermore, it is argued that, on account of governments being in a position to develop water powers much more cheaply than private individuals can, many of these which must inevitably go to waste if left to private initiative would be utilized to the great benefit of the people.

In reply to these theoretical arguments certain examples of government ownership may be cited and must be given considerable weight because they are examples of government ownership in actual practice and not merely deductive arguments based on theoretical considerations.

A classical example of government ownership is found in the case of the Chemin de fer de l'Ouest in France, which is well known to compare most unfavorably with the privately owned railways.

Perhaps, however, one of the best examples of the failure of government ownership is found in the British telegraph system. This was originally a private enterprise, but was taken over by the government in 1870. For the first two years, that is, during 1870 and 1871, the government made a small profit after paying interest on the debentures issued for the purchase of the system, but since that time the system has been run at a loss. In 1914 the Rt. Hon. Sir Charles Hobhouse, the Postmaster General, in a speech in the House of Commons, declared that in the last 40 years the British Government's expenditures on the telegraph system, not including interest on annual losses, nor any provision for amortization, were in excess of receipts by a sum of $110,000,000, while if the interest on losses and amortization were charged the loss would be $200,000,000. The average annual operating loss for the years 1908 to 1915, inclusive, amounted to

over two million dollars while for the same period the average loss including interest and new money expended amounted to more than five and a half million dollars. In 1916 a committee appointed by the government to investigate the possibility of saving money in government expenditures stated in regard to the telegraph system:

"The history of the telegraphs is most unsatisfactory. They were taken over in 1870 at a cost (including capital expenditures on extensions) of 10,129,687 pounds in the anticipation that they would yield a profit to the State. After the second year of Post office management the profit failed to cover interest on the capital outlay. Year by year the financial position has grown worse and worse. In recent years the loss upon working has not been less than 1,000,000 pounds a year and this loss includes nothing for interest due to the State upon the aggregate losses of previous years."

Another example of government ownership that is, perhaps, more interesting than the British telegraph system because it is found on this continent and because it is an undertaking initiated much more recently is the telephone system of the Province of Manitoba, Canada. It has been the subject of a most valuable and interesting study by Professor James Mavor, of the University of Toronto, published in book form by Moffat, Yard and Co. of New York, and the MacLean Publishing Co. of Toronto, under the title "Government Telephones, The Experience of Manitoba, Canada."

The Manitoba Government took over the telephones in 1907 and Professor Mavor's study covers the history of this experiment from that time till the year 1915. It would take too much space to review the book here, but we may quote one of the final conclusions amply borne out by Professor Mavor's masterly study:

"The entire history of the government telephone enterprise in Manitoba affords evidence of the most positive character against government ownership. Practically all of the defects which have emerged elsewhere in the management of industries by State officials have made their appearance in the case of the Manitoba Telephones. The management has been ineconomical, the enterprise has been handicapped by political intrigue, the finances, mingled as they have been with the general finances of the

Province, have been unsoundly administered from the beginning, and the obligations of the public have been enormously increased without adequate compensatory advantages.

"It is possible that only by repeated and costly failures such as the Manitoba Government Telephones, will the public realize that the proper function of government is not the conduct of industries, but the impartial inspection of them under intelligent laws adapted to the character and conditions of the community and the country."[2]

These are typical examples where we might expect to find confirmation of the arguments advanced by advocates of government ownership, but this is lacking.

As regards the government ownership of water powers we have fortunately one large-scale experiment in the case of the Hydro-Electric Commission of Ontario. This system is composed of several different parts throughout the Province of Ontario, Canada, and up to October 31, 1917, involved a total capital expenditure of $36,976,900.99.[3]

The first important agitation looking towards government ownership of water powers in Ontario apparently began in 1900 when the Board of Trade of the City of Toronto applied to the legislature for power to undertake the development and transmission of electricity from Niagara Falls.[4] This application was not favorably received, but the agitation in favor of public ownership continued and finally in January, 1906, the government appointed the Hydro-Electric Commission of Ontario. This commission was given very great power as may be seen from a study of the legislation which has been enacted and which may be found in a blue book entitled "The Power Commission Act and The Hydro-Electric Railway Act, etc.," published by A. T. Wilgress, Toronto, 1917. This blue book will in future be referred to as "The Power Commission Act."

Originally the Hydro-Electric Commission was supposed to engage merely in the transmission and distribution of electric current; but it soon determined to undertake the generation of electricity on its own account, to construct its own hydro-electric

[2] "Government Telephones," by James Mavor, p. 163-164.

[3] Hearings before the Committee on Water Powers of the House of Representatives, Sixty-fifth Congress, Part 3, p. 725, *et seq.*

[4] *Ibid*, page 702.

plants, to compete with private companies and there can be little doubt that it intends, if possible, to monopolize the hydro-electric systems of Ontario judging from the statement of its chairman Sir Adam Beck[5].

It would, of course, be impossible in this report to make anything approaching an adequate review of the results of the Hydro-Electric Commission's activities; but reference should be made to an extremely interesting study of these in a pamphlet entitled "Public Ownership and the Hydro-Electric Commission of Ontario," published by the MacLean Publishing Co., Ltd., Toronto, 1917. This contains a reprint of a series of articles by Professor James Mavor, of the University of Toronto, as well as certain other articles which appeared in "The Financial Post of Canada." Future references to this pamphlet will be abbreviated to "Public Ownership."

In his examination of the Hydro-Electric Commission Professor Mavor considers twelve objections to government ownership and illustrates these from the history of the Ontario scheme. These objections may be briefly set forth as follows:

1. Increase in political power is one of the main motives in public ownership.

2. The conduct of industrial enterprises by government officials is not subject to inspection by impartial authorities.

3. The management of publicly owned enterprises is in-economical.

4. In entering into industrial undertakings the government is apt to minimize the risks involved and to underestimate the amount of capital required.

5. Government officials are reluctant to provide for the continuity of the enterprise by setting aside adequate depreciation and reserve funds.

6. In governmental activities of this character there is a tendency to promote the illusion that profits inhere in industrial enterprises and to disregard the fact that, save in rare cases of adventitious profits, these are due to economy and skill in management.

[5] "Hearings before the Committee on Water Power."

7. The tendency to overman governmental enterprises of this character and the engagement of employes on political rather than technical grounds.

8. The tendency to fix the price of the government product at an arbitrary amount so as to induce the public to believe that the service is being rendered cheaply, rather than at a rate determined by the technical conditions of the enterprise.

9. The tendency toward frequent crises in the management of the enterprise, resulting from the fundamental unsoundness of the methods usually adopted.

10. The absence in governmental industrial enterprise of an experienced board of directors, its place being taken by a committee of politicians or the nominees of politicians.

11. A government preoccupied with industrial enterprises neglects its proper functions.

12. Government ownership tends to promote the illusion that politics and business are interchangeable terms[6].

It will be seen that this treatment of the subject of government ownership of power plants is a consideration thereof on the broad ground of public policy, but it may be well to look for a moment in more detail on the immediate effect on electrochemical industry.

The one great argument advanced by the advocates of government ownership is that of the low price of power. There can be no question that the prices charged for domestic use by the Hydro-Electric Commission are, at least in many cases, apparently much lower than could formerly be obtained. This apparently lower rate can be reached for various reasons. For example, the Hydro-Electric Commission does not pay taxes as a private corporation must. Then by a peculiar application of the principle of "no discrimination" it is possible to charge an extremely small rate for domestic purposes. According to Sir Adam Beck: "The small user buys electricity at the same price as the large user. There is a standard rate in every community which applies whether you use 10 horsepower or 10,000 horsepower. In the case of domestic users the small consumer buys it for less than the large consumer."[7]

[6] "Public Ownership," pages 15-48.

[7] "Hearings before the Committee on Water Power," page 714.

When the actual rates charged domestic consumers are examined it is found that "no discrimination" means frequently that the consideration of the cost of the service is omitted. Thus on the Niagara Falls system it is found that the domestic rates for Niagara Falls, Toronto, Kitchener, Hamilton, and London, are the same though the current is generated in Niagara Falls. Now, it is obvious that the cost of the current at a place like London, more than 100 miles from Niagara Falls, must be more than at Niagara Falls where the current is generated, yet the price charged for it is the same. Then we have in Kitchener, about 80 miles distant, Hamilton 40 miles, Toronto 80 miles, the same price.[8]

In view of such an interpretation of "no discrimination" it is not surprising that a movement is now on foot among the consumers to agitate for a "uniform rate" which means that all customers of the Hydro-Electric Commission shall pay the same rate irrespective of their distance from the place where the power is generated.

It is obvious that the cost of power to consumers in the government-owned hydro-electric system is largely governed by their voting power and under such conditions electrochemical industries are bound to suffer since they will certainly have to bear more than their fair share of the actual cost. It therefore follows that government ownership, so far as electrochemical industries is concerned, is not desirable.

Another point that should be noticed is that, in spite of the extremely favorable conditions, such as freedom from taxation, that the government-owned system enjoys, private companies are still able to compete successfully with the government enterprise.

No one will deny that there have been many reasons for complaining of the behavior of private companies, but the evils thus resulting can be remedied through government regulation based on the principle of seeing that justice is done to all; but when the government itself engages in business enterprises of this sort there is no hope of getting justice and herein lies the greatest danger of the government ownership of water powers. This is admirably illustrated in the case of the Ontario Hydro-Electric Commission.

[8] Tenth Annual Report of the Hydro-Electric Power Commission, Vol. II, p. 164, *et seq.*

The Government of Ontario, through the Commissioners of the Niagara Falls Victoria Park, made a contract with the Electrical Development Company of Ontario to rent water power to it and under clause 16 of the agreement of January 29, 1903, bound itself as follows:

"The Commissioners will not themselves engage in making use of the water to generate electric, pneumatic or other power excepting for the purpose of the park."[9]

This agreement, of course, stood in the way of the Hydro-Electric Commission when it undertook to generate power from the Niagara River in 1916, accordingly the following remarkable legislation is recorded in The Ontario Niagara Development Act:

"The exercise of the powers, which may be conferred by or under the authority of this act or of any of them, shall not be deemed to be making use of the waters of the Niagara River to generate electric or pneumatic power within the meaning of any stipulation or condition contained in any agreement entered into by the Commissioners for the Queen Victoria Niagara Falls Park. (6 Geo. V., ch. 29, cl. 7).[10]

Naturally enough the Electrical Development Company was disposed to fight the government's repudiation of its agreement and therefore attempted to have the legality of its original contract passed on by the courts. This, however, proved to be impossible because the Hydro-Electric Commission had procured legislation of this character:

"Without the consent of the Attorney-General, no action shall be brought against the Commission or against any member thereof for anything done or omitted in the exercise of his office." (6 Edw. VII, c. 15, s. 21.)[11]

Since of the three men constituting the Hydro-Electric Commission one of them was the Attorney-General it is not surprising that the fiat which would permit the testing of the legality of the Electrical Development Company's contract was refused.

The Hydro-Electric Commission also has extraordinary powers in regard to the making of regulations as to the design, construction, protection, operation, maintenance and inspection of all elec-

[9] "Public Ownership," page 7.
[10] "The Power Commission Act," p. 176.
[11] *Ibid*, p. 22.

trical apparatus whatsoever used in the Province of Ontario, may order compliance with these and may order power supply cut off pending such compliance.[12]

A very serious situation was created by the Hydro-Electric Commission's activities during the war and electrochemical industries were the chief sufferers. In spite of the serious shortage of power the Commission made every endeavor to extend the use of electric power in directions which would give it political power, with the result that electrochemical industries of great importance, not only to the prosecution of the war but to normal activities, were seriously hampered by having their power supply cut off.

Recapitulation.

Whatever effect, whether good or evil, government ownership of water powers has on electrochemical industry, will be transmitted to the nation at large.

Arguments advanced by advocates of government ownership are generally *a priori* and not based on actual experience.

Examples of governmental activities in various fields which can be undertaken by private enterprise offer evidence strongly against public ownership in such cases.

An examination of the only adequate example of government ownership of a hydro-electric system shows that all the evils, associated with governmental activities in fields other than those for which governments are instituted among men, have developed, and that the largest individual users of electric energy, the electrochemical industries, will be the most direct sufferers therefrom.

Additional Note.

Since the above was written some more interesting and important information in regard to the Ontario Hydro-Electric Commission has become available in the shape of the report of the Messrs. Clarkson & Sons, chartered accountants, on the accounts of the commission. The government promised to have this investigation made in 1916,[13] but the report was not apparently available to the public till March, 1918, and had evidently been

[12] *Ibid,* p. 41, 42.
[13] "The Financial Post," Toronto, March 3, 1917.

kept from publication for a considerable time as it only covers the period ending October 31, 1917.

Apparently the report, in spite of its inadequate nature, is in accordance with what might be expected in a governmental enterprise of this kind. For example, the original Hydro Act provides in regard to *payment* on sinking fund account that "a municipal corporation . . . may be relieved by the Commission from payment of any sum on account of the sinking fund account for the first five years . . . the amounts . . . on sinking fund account shall be payable during the next ensuing thirty years."[14]

This has been interpreted as meaning that no municipality shall be charged any sinking fund for five years.[15] This ingenious distortion of the meaning of the act permits municipalities to conceal an actual loss of their enterprise.

It also appears that the report covers only the provincial accounts leaving out the systems of the municipalities which are controlled by the Commission.[16]

One remarkable charge shown by the report is a sum of $1,117,433.86 against the rate payers of the Province, a large number of whom are in no way benefitted by the Commission's enterprise. This money is for items such as "Investigations and reports on proposed municipal railways," "Demonstrations at exhibitions as to use of electricity on farms." "Engineering assistance and estimates to municipalities not under contract," etc.[17]

This charge should, of course, be against the municipalities included in the Hydro combination or against such of these as were directly concerned; but had this been done the Government's enterprise instead of showing the net surplus of $174,919.31, which it is made to do, would show a net deficit of $942,514.55.

Obviously this report shows that had the commission kept "regular accounts" as required by the act[18] "the results of an audit would show, and this report should show, that the operations have been carried on at considerably less than cost.[19]

Thus as time goes on the evidence which accumulates confirms

[14] "The Power Commission Act," page 30.
[15] "The Financial Post," Toronto, March 15, 1919.
[16] *Ibid.*
[17] *Ibid.*
[18] "The Power Commission Act," page 6.
[19] "The Financial Post," Toronto, March 15, 1919.

what has already been observed in a large-scale experiment on government ownership of hydro-electric systems. It is only too obvious that those most directly affected will be the largest users of electric power, the electrochemical industries, and, looking at the subject from the broader aspect of the general good, that any partial or transient benefit which may be derived from this encroachment of government upon industrial fields will be greatly overbalanced by permanent evil.

DISCUSSION.[1]

CARL HERING:[2] As I declined to sign that report as a member of the committee, I would like to state my reasons. I did not sign it because I think the report is unbecoming to this Society and not a creditable one; it is so evidently partial that, in my opinion, it loses much of its weight. It is not of a judicial nature, which I think it ought to be, coming from a Society like this. It is a report such as an attorney on one side of a case may make, and those of us who have been in court, know that when we hear the second attorney make his argument, we often wonder why we ever believed what the first attorney said. I think it is the proper function of this Society to act in the capacity of an impartial judge who weighs both sides carefully.

The report also omits entirely any reference to an illustration of existing government ownership and operation in this country, namely the post office and the parcel post. This is owned and controlled entirely by the government. I think there are very few in this room who would agree to let the mail and parcel post system go back to private corporations to be conducted for profit. If we did, we would probably soon have to again pay 25 cents a letter, as used to be the case before the government took control of the mails.

[1] Before the preceding report and its appendix was presented, President F. J. Tone made the following introductory remarks: At our meeting in Atlantic City, the Society instructed the Public Relations Committee to take up the consideration of the question of Government ownership of water powers and to report at this meeting. A report has been prepared which has been approved by the Board of Directors, and I am going to ask Mr. FitzGerald, who has so ably prepared this report, to present it for your consideration.

[2] Consulting Electrical Engineer, Philadelphia, Pa.

Nor does the report refer to successful cases of government control and operation in other countries. I understand that in the city of Glasgow, for instance, where government control is carried to an extreme, the people are relieved entirely from taxes on account of the economic operation due to the government control, under which these industries become self-supporting, like our post office. I understand, too, that in Sweden the government controls the water powers for the benefit of the people. These cases are not even mentioned, nor is there any reference made to Cleveland or other cases in this country where government control has benefited the people greatly.

What we want as electrochemical engineers is cheap water power. Now one way in which it *could* be gotten is through government ownership and management, *provided* it was managed properly. The objection to government management in this country is the corrupt and degenerate political system we have here, and it seems to me that this is what we should attack and not the fundamental principle of government management in general. There are two kinds of opponents to government management in this country; one, those who believe that government management would be good *if* politics could be eliminated; the other kind are the financial interests who want to profit from water powers. Now it seems to me that we should, in our report, distinguish properly between these two. Are we to be the cat's paw of the capitalists who wish to profit from water powers, or do we object to government control simply and solely because politics enters too much, so that the management cannot do what it ought to do? It seems to me, therefore, that we ought to say clearly in our report which of those two positions we take; we should have no camouflage report; and I think it would be the beginning of the downfall of this Society if we should take the part of the financiers to aid in their profiteering at the expense of the electrochemical industries.

Therefore I move that the report be sent back to the committee to be revised so as to state these issues clearly, and show clearly which side we are going to take; whether we are defending the capitalists who want to profiteer from water power, or whether we object to government control simply because, under the present circumstances the management is bad. Moreover, to take a vote

at this meeting, where there are present perhaps three percent of the members of this Society, would mean that a little over one percent would carry the motion; I do not think that this Society should go on record as endorsing something which is voted for by only three percent of its members. One way to cure a corn is to cut off the foot, but that is not a very wise way to do it. We should try to find out just where the trouble lies and then cure that trouble. In other words, we should not deprive ourselves of the possible benefits of government control of water powers simply because in this country corrupt politics would be likely to enter. Let us locate the real disease and try to cure it, but not kill the goose that could lay the golden egg, if cured of its disease.

F. A. LIDBURY:[3] May I ask what Dr. Hering's motion is?

CARL HERING: That the report be sent back to the committee to be revised, to state clearly which of the two positions we take, as to whether we are supporting the financial interests who want to profit from the water powers, or whether we object to government control at present simply because politics enters and the management is therefore generally bad. In other words, if government management and operation could be improved and corrected, then we would approve it.

F. J. TONE:[4] Is Dr. Hering's motion seconded?

CARL HERING: To make the motion brief, it is that the report be referred back to the committee for revision after considering this discussion.

H. W. GILLETT:[5] I am going to second Dr. Hering's motion. I agree with the report of the committee, but I also agree with Dr. Hering's comments on the wording. I think it should be more judicial; I second the amendment.

F. A. LIDBURY: I would like to ask Dr. Hering a question which I hope he will be able to reply yes or no to, and that is this, does he believe that in view of the history of public ownership, in view of the history of politics, in view of what is taking place today and in view of what he knows of human nature, that government ownership and operation of water powers could be at the

[3] Works Manager, Oldbury Electrochemical Co., Niagara Falls, N. Y.
[4] President, Carborundum Co., Niagara Falls, N. Y.
[5] U. S. Bureau of Mines, Ithaca, N. Y.

present time, under present conditions in this country or anything we are likely to see in his lifetime or mine, a success? If he will define his position on that, why, we will know how to talk to him. Just at present I am left very much in the dark as to whether he merely has a theoretical idea that government ownership would be a mighty fine thing if it were not so rotten, or whether he thinks it is practical.

Carl Hering: I can answer this first part with yes, that is, I think that reform in government *should be* and *is* possible, if we direct our attention to such reforms. Does that answer your question? I am optimistic enough to think human nature is such that it is possible to reform corruption.

F. A. Lidbury: Dr. Hering claims that the report as laid before this meeting does not state whether the committee is in favor of the point of view of financiers. There is no mention in the report of financiers, and the committee, in considering the matter, have not bothered themselves about the point of view of financiers or anybody else except the point of view of the electrochemical industry. The statements that are made in the report are based upon a practical consideration of facts, and the committee has, in my opinion, in presenting this report, avoided going into debatable and theoretical questions that Dr. Hering now wishes to draw, like a red herring, across the very practical trail that the committee has laid down for discussion at this meeting. It is the kind of political dodge that we would expect of an advocate of government ownership and operation, and I say that this meeting should not countenance any attempt to becloud the main issue, which is a scientific, engineering and technical one, such as this Society can properly discuss. The only reason, apparently, that Dr. Hering can give for wishing to see government operation is that he thinks it has been a success in the case of the post office and parcel post. I presume that recent evidence of the operation of governmental activities in the railroads and in the telegraph and telephone systems will confirm him in that opinion; but in that connection I would like, with your permission, Sir, to read a letter that was signed by one of the members of the Public Relations Committee, Dr. Carhart, which addresses itself to that particular phase of the subject:

"I heartily endorse the report of Mr. FitzGerald on Governmental Operation of Hydroelectric Plants. To sustain this report we need only point to the inefficiency and bungling of the Federal operation of the railways, the telegraph lines and telegraph communication. The cost to the public has increased to a very appreciable extent, while millions of dollars must be paid out of taxation proceeds. Nothing could be more convincing as to the inefficiency of Federal operation of public utilities than the Postmaster General's bungling and autocratic handling of telegraphic and telephonic operation. Night letters are delivered by the Post Office, in many cases hours later than under private operation, and long distance calls in most cases are more expensive. I see no advantage in Federal control and operation in times of peace. It is better in every way to leave it to the enterprise and push of private companies."

Yours truly,
(Signed) HENRY S. CARHART."

Now, Sir, I think we are entitled to a little more than a statement from Dr. Hering, that, because he does not like the conclusions of the report, the report is biased and that it does not consider something that I think, in the opinion of those present in the room, it ought not to consider, namely, the political aspect of government ownership as distinct from the effects of government ownership, and before anyone who may be inclined to vote for Dr. Hering's motion does so, I would ask him to take the report as it stands and look at it and read it or ask for it to be read again, and attempt to point out where, in that report, there is anything objectionable.

CARL HERING: May I refer to one more point which I omitted? We are too apt to judge the government management of today from the immediate past. I grant that the mails and the railroads and everything else of that kind which the Government has done in the immediate past, was very wretchedly done, there is no question about that, but that was under war conditions. We are now talking about peace conditions, we must therefore consider the management under peace conditions, and surely the management of the post office and parcel post under peace conditions was not as expensive, decidedly not as expensive, as it would be under private corporations conducted for profit.

H. W. GILLETT: I want the meeting to understand my object in seconding Dr. Hering's motion. My object in seconding that motion referred purely to the wording of the report. I agree heartily with the conclusions of the committee, but I think that

report would have more force if it were put up in a more impartial manner. It does have the earmarks of a piece of special pleading, sound as the arguments therein are. It seems to me that when you have got a good cause, you do not have to put it up as a matter of special pleading, you can rest on the facts, and so my objection is solely based on the language and the general flavor of the thing rather than anything else.

M. J. Brown:[6] I agree very heartily with Mr. Gillett's remarks. Unfortunately I have not been cognizant of the main facts regarding the source of the request for this report nor its destination. I presume that if submitted to and adopted at the meeting today, it will thereupon be released as the findings of the Society. I do not think it is quite fair to a committee that has done good work to throw a report of this kind back to it without very specific suggestions as to what changes should be made. There is one point that came to my mind as this report was read, for the sake of which I should like to have a chance to read the report over, privately at my leisure, so that I could acquire clearer ideas. Mention is made, for example, of the fact that London, Ont., gets power at the same rate as points that are nearer Niagara Falls. We have had some findings of various legal bodies in recent months in some of our States which throw a side light upon that situation. For example, there is one of our States wherein the Public Utilities Commission has ruled that all contracts for power given to consumers of domestic power or wholesale power are of no force if subsequent to their execution the Commission chooses by general rules to overrule the individual contracts. These matters have not been carried through the courts; they may get to the Supreme Court only in an indirect manner, but the fact is that permission to contract out power in large amounts at low rates has been taken away from certain power producers in certain of our States, and there is at least a 50 percent chance that these rulings will stand. Now we presume, I think, most of us, most of the time, that the producers of power, whether it be Niagara power or other power, are going to be given the permission to contract out power at whatsoever rates they deem proper under private management. That is a condition of affairs

[6] The Roessler and Hasslacher Chem. Co., Perth Amboy, N. J.

which may not be allowed to prevail. We already have had an expression of the opinion of the people in this country quite generally over the proposition to raise mail rates so that mail going a great distance can be charged more than mail going a short distance. I am of the opinion that the majority of the people of this country are against that; in other words, they will favor sending mail from New York to San Francisco at the same rate they do from New York to Philadelphia, and there is some reason for such a ruling, and it may be that the people of this country will decide in the not very far distant future that Niagara power belongs no more to the people of Buffalo than it does to the people of Syracuse, and that the means for developing and distributing that power should be borne by the State or the Nation, so that the cost may fall impartially upon the users wheresoever they may be. Now I throw that out as one suggestion, and I think that the points I have mentioned rather bear out Mr. Gillett's suggestion that this report does sound like a case of special pleading. Personally, I should welcome the opportunity to read this report over with the further opportunity of voting upon it, say by mail, or expressing my opinion in some other meeting. I do not think that the report should be thrown back to the committee, which has undoubtedly done very good work, without very intelligent and specific suggestion of the changes that should be made.

F. A. J. FitzGerald: To save further discussion on that point, I am afraid I did not bring out clearly enough that all that evidence as regards hydroelectric conditions, etc., is not part of the report, it is partly an appendix; the report itself has no reference to the hydroelectric condition, it is just the appendix.

H. W. Gillett: What is the relation of the appendix to the report? Is the appendix merely the committee's reasons expressed here for making the report? That is, when we vote for this report, do we have to approve the appendix as presented, or are we dealing with the report as just read?

F. A. J. FitzGerald: With the report. The responsibility for the appendix is solely mine.

(Mr. Gillett then withdrew his seconding of Dr. Hering's motion.)

L. E. SAUNDERS:[7] I move that the report of the committee be adopted, that it be printed in the Transactions, and that a thousand extra copies be printed as soon as possible for distribution at the discretion of the committee.

J. W. RICHARDS:[8] I think that the appendix should be simply published as a paper contributed to the Society on this general subject for which only Mr. FitzGerald is responsible.

WILLIAM BLUM:[9] Speaking to Mr. Saunders' motion, I would like to raise the question from the standpoint of the policy of this Society, that inasmuch as this is a scientific society and not a commercial or political organization, is it desirable for the Society to go on record on the public ownership question which is so closely tied up with political questions and on which, therefore, it is not probable there will be unanimity of opinion throughout the Society?

F. A. LIDBURY: In answer to Mr. Blum's remarks, I think this is one of the occasions on which it is not only desirable for the Society to go on record, but I think that the Society owes it to itself and the industry it represents, as has been very clearly brought out in Mr. FitzGerald's paper. One of the absurd results of the government operation of water power is that the electrochemical industries, on account of their relatively small voting power that they may possess, get discriminated against in a matter which is essential to their existence, and unless the only body I know of in this country that is able and willing to put their point of view before the public and before legislative bodies, has the courage of its convictions and the willingness to back them up, why we have only got ourselves to blame if these things go by default. I think it has been pointed out in successive presidential addresses that a good many things that we all complain of in legislative enactments which are not to the good either of the industry that we represent or the people at large, simply go through because we take the point of view that Mr. Blum suggests we now take on this matter; in other words, we say that we are individuals rather than a society which represents, more than anything else in the country, a certain specific and very useful industry, and

[7] Manager, Abrasive Plants, Norton Co., Niagara Falls, N. Y.
[8] Professor of Metallurgy, Lehigh University, Bethlehem, Pa.
[9] Chemist, Bureau of Standards, Washington, D. C.

my own opinion very strongly is that as long as there is substantial unanimity, I would not want to have a question like this be put forward as the view of the Society, if it did not represent substantial unanimity; but as long as a vote has determined that substantial unanimity on this subject prevailed in the Society, I feel it is not only our privilege but our duty to go on record and to push our views, not only as a society but as individuals, wherever and whenever we get an opportunity to do so.

CARL HERING: I wish to amend Mr. Saunders' motion, to table the matter.

(The amendment was seconded and lost.)

J. W. RICHARDS: May I make this remark, that although we have a small percentage of the total membership of the Society, yet the report as it is before us for adoption is agreed to by ten of the past presidents of the Society out of thirteen, two not being heard from and one opposing the report, so that it is a matter which has been carefully considered by them, and, therefore, the vote is practically not simply the vote of a small percentage of the Society, but is confirming the opinion of the Committee on Public Relations composed of most of the past presidents of the Society.

F. J. TONE: The question is on the adoption and publication of the report.

(The report was then adopted, the vote being 45 to 6.)

CARL HERING: I wish to get on record that this vote was taken at a meeting at which less than 3 percent of the members of the Society were present.

F. A. LIDBURY: Might I ask Dr. Hering, in view of the record he has made, if he is desirous and willing to move that the vote at this meeting be neglected and that a referendum of the Society on the subject be taken? I will be glad to second that motion if he will make it.

C. G. SCHLUEDERBERG:[10] I think in view of the discussion, it would be a very good idea to refer this to the entire Society, and give all the members a chance to know what is going on and what this report is like. The report on its final adoption will gain considerable prestige thereby.

[10] Electrical Engineer, Westinghouse Electric and Mfg. Co., East Pittsburgh, Pa.

F. A. LIDBURY: That is perfectly true, but it seems to me that if members do not take sufficient interest in the affairs of this Society to attend the meetings and make themselves heard on matters of this kind, they can hardy expect to be consulted every time the Society takes action. We have had at this meeting an expression of opinion of 45 to 6 of those present and that indicates, as well as the fact that 10 to 1 of the members of the Public Relations Committee signed the report, in my opinion, what the opinion of the Society is on the subject. I am perfectly willing that there should be a referendum of the subject, and I will be glad to vote for it, but I think the demand should come from the opponents of this report; if they have the courage to ask for that referendum, I will be glad to uphold it.

C. G. SCHLUEDERBERG: 45 and 6 make 51, and there are almost 2,000 members in the Society; the percentage voting is manifestly extremely small. In regard to the members coming to the meetings, of course they ought to come, I would like to see them all come.

T. F. BAILY[11]: Is there not about as many voting here today as usually vote at the annual election?

C. G. SCHLUEDERBERG: No.

CARL HERING: Decidedly not, no.

F. J. TONE: Mr. Lidbury, did you make a motion to reconsider?

F. A. LIDBURY: No, I asked Dr. Hering, in view of the objection he noted, whether he wished to move for a reconsideration and referendum to be taken, in which case I offered to second his motion. If he does not want to do that, if he does not think that he would stand any better chance with a referendum, why should we go to the trouble of taking one?

CARL HERING: It seems useless to me to make any more motions, but I would like to say that if this is to be referred to the Society at large, the discussion should go with it, not merely a short, brief summary, but at least an abstract of the discussion for and against, and a history of the vote. There is no need of

[11]General Manager, Baily Electric Furnace Co., Alliance, Ohio.

making a motion to reconsider because no action has been taken to refer it to the Society at large.

F. A. LIDBURY: I will make a motion to reconsider, that the report be circulated among the membership, accompanied by Mr. FitzGerald's paper and also accompanied by any statement of the opposite side that Dr. Hering wishes to make. I see no reason why the rather rambling discussion, we have had here, a lot of which was not to the point of the subject, should be circulated among the membership for this purpose.

CARL HERING: I do think that an abstract of the discussion and a history of this action should go to the absent members; it is not fair to them not to give it.

(The motion to reconsider was seconded.)

F. J. TONE: Is there any further discussion of this motion to reconsider and take a referendum vote, to submit to all the membership the report with the appendix and with any other discussion it may be desirable to add?

S. N. CASTLE:[12] I think Dr. Hering should put before the membership the statistics showing the financial benefit to the country, showing whether the Government makes a profit by the operation of the post office or the operation of the railroads, all of which he claims in his impassioned appeal. In other words, the committee have given us definite figures while Dr. Hering has only talked generalities and drawn certain conclusions which are shown to be entirely erroneous by the figures submitted.

F. J. TONE: The question is on the reconsideration of this report and submitting it to a referendum vote.

CARL HERING: The reconsideration of the vote to adopt the report?

F. J. TONE: Yes, and having reconsidered it, I suppose the balance of the motion is in order.

J. W. RICHARDS: If this is put to a referendum vote of the Society, I think the vote here is typical, that probably in the ratio of seven or eight to one the members are going to vote against government ownership. I think it is rather unnecessary to take

[12] General Electric Co., New York City.

it to the whole Society, and I again call attention to the fact that the majority of the past presidents of the Society are in favor of it. I think the motion is entirely unnecessary.

F. J. TONE: I think these motions had better be separated and a plain motion to reconsider would be more parliamentary.

F. A. LIDBURY: I would not want to make a motion to reconsider unless it is going to be referred to the Society; that is the only object for moving its reconsideration. If the first were carried and the second lost, there would be no object in reconsidering the vote.

F. J. TONE: The motion is to reconsider the motion adopting this report and to submit it to a referendum vote.

(The motion to reconsider was lost.)

F. A. J. FITZGERALD (*Communicated*): After reading the report of the Committee on Public Relations, I explained that I would read a paper which I had prepared on the subject of the public ownership of water powers. This evidently was not understood by several of the members present, hence the discussion. It may be well to explain why the paper was written. In preparing the rough draft of the report for the committee, the draft which was subsequently used in part for making the report, I found that there was a great quantity of available material which was published in support of public ownership of water powers, but relatively little that was opposed to it, particularly as regards electrochemical industry. It, therefore, seemed desirable to prepare a paper which would set forth some of the arguments against public ownership of waterpowers and particularly to examine the one really large scale experiment in public ownership of waterpowers of which some details were available. I thought it possible that this paper might be a useful appendix to the report; but, of course, the Public Relations Committee were in no wise responsible for it.

A paper presented at a meeting of the Pittsburgh Section, February 14, 1919, and called up for further discussion at the Thirty-fifth General Meeting of the American Electrochemical Society in New York City, April 3, 1919, President Tone in the Chair.

POWER PRODUCTION FOR ELECTROCHEMICAL PURPOSES.

By C. S. Cook.[1]

Abstract.

A discussion of the cost of steam power in the Pittsburgh district, particularly when generated in large stations and when off-peak power is used for electrochemical and electrometallurgical purposes. The author concludes that this kind of steam power can compete with water power in all except the most favorable cases for developing hydraulic power.—[J. W. R.]

Under normal conditions perhaps no part of the country is in better shape to supply steam power to electrochemical industries than is the Pittsburgh district. Normally our coal costs are low, river water conditions are fair, together with a fair labor market.

The importance of cheap fuel is self evident. Few, however, realize the emormous amount of water required for condensation purposes by the modern turbine plant. A 200,000 kilowatt plant requires, in round figures, 1,000 cubic feet of water, or approximately 8,000 gallons of water per second at a temperature of 75 degrees, for condensation purposes. This amounts to 28,000,000 odd gallons per hour, or assuming full load, something like 675,000,000 gallons per day of twenty-four hours. This should make it readily manifest that large steam-driven power stations can only be located on either the seacoast, the lakes or on large rivers, where such quantities of water can be readily obtainable.

During the past few years the change of conditions brought about by the war have greatly increased the price of fuel, coal, and also largely increased the price of labor, which are the two items most largely forming the operating cost of the modern

[1] General Manager, Duquesne Light Co., Pittsburgh, Pa.

steam power house. In the rates for labor and coal which now obtain even in the Pittsburgh district, there is frankly no use of the electrochemist expecting rates of power in any way comparing with those which obtain in a few of the larger hydraulic installations in the country, some more notably perhaps than others—Niagara Falls—where the Lord has furnished the water, built the dam, and where the investment per kilowatt is reasonably low and the maintenance and renewal and replacement charges of operation almost negligible. However, there are other things which the electrochemist or any other user of power must consider in the establishment and operation of his business, such as the freights on raw material from the points of their production to his location, and the freight on his manufactured product, which is frequently the raw material employed by other manufacturers, from his plant to destination. For heavy or bulky materials, these freights may at times offset in a large measure the difference in cost between steam-driven and hydraulic-electric power. These matters should be thoroughly considered before the establishment of any kind of industry—chemical or otherwise—employing power in large quantities. It is perfectly axiomatic that power must be obtainable at a price which will enable the user to employ it in the production of his product at a profit which will justify the capital expenditure required in his business. It is also just as axiomatic that the utility company furnishing it, whether steam-driven or hydro-electric energy is delivered, must receive for that energy a price which will cover its cost of production and a just return upon the capital invested in the facilities necessary for such production. Such just return must not only cover a fair rate of interest even in times of business depression but must also cover an amount adequate for the current maintenance, insurance, taxes, operating exigencies, such as storm damage, etc., together with an amount sufficient to cover the renewals and replacements of the apparatus as it may in time become worn out or its retirement be made necessary through obsolescence.

Many cases have come to our attention where manufacturers and others claimed they could produce power at exceedingly low rates, where we found that their estimate of cost simply constituted what labor and fuel were necessary, without any consider-

ation of the thousand and one other expenses or overhead items which, like death and taxes, are unavoidable.

Power production on that system of costs invariably leads to financial difficulties; and if the amount and cost of power is a large item in the cost of the manufactured article produced, serious financial troubles are of course liable to occur.

There is one thing that we have been impressed with in discourses with people who are interested in the manufacture of electric steel, that is, who are operating steel furnaces. Frankly, many of them do not seem to know how to sell their own product to the best advantage. Now this is perhaps rather a broad, bald statement, but we happen to know one manufacturer whose business requires large quantities of steel castings, some of which are very large and of difficult shape to cast. The ordinary open-hearth castings, costing under normal conditions of normal times about 4½ cents per pound, are made easily 20 percent heavier than would be required were the higher grade, smoother and more homogeneous casting that the electric furnace produces used in their stead. Again, the open-hearth casting as brought into this manufacturer's works requires a very considerable expense in welding and acetylene cutting before they can be incorporated into the machines which the manufacturer produces. That manufacturer, we are certain, could well afford to pay 5½ cents for electric-furnace castings in normal times if the matter were properly put up to him through proper channels, rather than 4½ cents for normal open-hearth castings. The only question we have is: Why do not some of the steel people see this condition and get busy on the problem? Not only would they be able to compete with open-hearth steel on this basis, but they could obtain for their castings a per-pound rate which would cover to them more than the entire cost of the electrical energy used in the production, even though it were purchased at the long-hour continuous rates of the average central station.

There is, however, with the steam-driven stations supplying power for general purposes an opportunity for lower rates or costs of power than those given by the continuous schedule, that is, through the employment of energy on the off-peak schedule, or energy provided for with a certain period of curtailment of

service either in whole or in part during certain hours out of each day throughout the business days of the week. This form of schedule is very common with the steam-driven utilities. No utility company has a constant load for twenty-four hours; in fact, few utilities, or electric-power companies particularly, have a maximum load for a space longer than two or three hours. Generally rates have to be made on such a basis that the class of customer demanding this short hour load bears the overhead which is inevitable in preparing for and carrying such short-hour load. The off-peak schedule, therefore, which provides for a curtailment of power to the customer, electrochemical or manufacturing, who can use power for long hours and can arrange for such curtailment during the peak period, can be made mutually advantageous; first, to the strictly peak customer, by diminishing somewhat the overhead burden that it is necessary to carry on his service; second, to the electrochemical customer, by establishing for him a lower rate due to the fact that he can be relieved and is relieved on such off-peak schedule of a large part of the overhead which follows a continuous service schedule; third, to the power company itself by establishing a condition of longer hour service and greater output during the periods when the system would naturally be underloaded, thereby reducing operating costs.

If power is to be applied to the electrochemical industries from steam-driven plants, that power must necessarily be supplied from the largest type of central stations. A careful study was recently made by the Hydro-electric Power Commission of Ontario on the rate of coal consumption in various electric generating stations and industrial establishments in Canada and the United States, and formed the subject of a report made by them on February 15, 1918. This report very clearly indicates the desirability, both from the standpoint of the utility company, the power user, and of the public generally, of large central-station units for the supply of power for industries generally. The consumption of fuel through the use of large central stations, with their attendant large boilers and turbines, alone justifies perhaps

the gradual elimination of the smaller plants throughout the country. To illustrate:

The average coal consumption per kilowatt year in stations of 650 kilowatts capacity is shown by the Canadian Hydro-electric Commission to be				34.6	tons	per kw.	Year
A station of 2,980 kilowatts capacity				19.0	"	"	"
7,230	"	"		17.8	"	"	"
24,600	"	"		12.75	"	"	"
96,000	"	"		8.8	"	"	"
149,000	"	"		8.4	"	"	"

The average load-factors of the stations analyzed, however, showed a gradual increase in the load-factor from 29.3 percent for the smaller station to 44.7 percent for the larger station, brought about of course by the greater diversity of the load served. When the hours of use are considered it will be apparent that the relative fuel economy between the central station power plant of a capacity of, say 150,000 kilowatts, as compared with the utility or smaller central station plant of, say 1,000 kilowatts, is something in the ratio of 5:1, a fact which in itself alone would determine that the future steam-power development of the country will consist of large central stations located along the seashore, lakes or rivers with jointly inter-connected distributing systems for mutual support and assistance. Such a system, or rather combination of steam generating systems, should be, and we believe eventually will be, able to fully meet the great and increasing requirements of electrochemistry and electrometallurgy.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

PRINCIPLES OF INDUCTIVE HEATING WITH HIGH FREQUENCY CURRENTS.[1]

By E. F. Northrup

PART I: THEORY

INTRODUCTION.

In the development of the electric induction furnace the invariable practice heretofore has been to employ electric currents of moderate or very low frequency. In all the forms which such furnaces assume they consist essentially of an iron-cored transformer the secondary of which is a single ring or loop of a conducting material which is to be heated. The iron core of this transformer passes through and so interlinks with its single-turn secondary. The iron core constitutes the magnetic circuit and the single-turn secondary one of the two electric circuits of the transformer. By interlinking both primary and secondary circuits with an iron magnetic circuit the necessary close coupling between the two is secured.

The primary winding of an induction furnace of this established type will possess self induction, the effective value of which will increase as the closeness of coupling between the primary and the single-turn closed secondary is diminished. Consequently the frequency of the e.m.f. impressed upon the primary winding must be maintained low to avoid an excessive lag of the primary current behind the impressed e.m.f. To obtain a good power factor frequencies as low as five cycles per second have been commercially employed. By the very circumstance of the use of low frequency, the employment of negative reactance to correct the power factor is precluded by the enormous capacity which would be required.

[1] Manuscript received January 22, 1919.

The possibilities offered for efficient inductive heating by going to the opposite extreme, of employing very high frequencies and dispensing with all iron magnetic circuits, does not heretofore appear to have been appreciated. The demonstration of the feasibility and the good efficiency of inductive heating without iron and with high frequency currents seems, therefore, to be new. The theory of this method of electric heating and an account of methods employed and actual results obtained deserve, it would seem, a clear and full presentation, which it is the object of this paper to give.

The writer's attention was turned to researches in this direction in July, 1916.

The experimental apparatus employed was constructed by the *Pyrolectric Instrument Co.,* of Trenton, N. J. The tests and experiments were made in and with the facilities of the *Palmer Physical Laboratory of Princeton University,* Princeton, N. J., and the work was done for and was financed by the *Ajax Metal Company,* of Philadelphia, Pa. The writer is particularly indebted to Mr. G. H. Clamer, vice-president of this company, for his continued faith in and unfaltering support of an enterprise which in its early stages appeared chimerical and presented seemingly insurmountable obstacles. To his patronage the final success of this new and valuable method of electric heating is in very large measure due.

ELECTRODE AND ELECTRODELESS HEATING.

In Fig. 1 are represented a primary circuit I and a secondary circuit II between which there is a coefficient of mutual induction M. L_1 and L_2 are coefficients of self induction, R and S are resistances, and a and b are currents. An e.m.f. E impressed on circuit I develops heat a^2R in the primary circuit and b^2S in the secondary circuit. When the heat so developed electrically is used for the production of temperature throughout an inclosure or for the heat treatment of some material, we have a device which, in the broad sense, constitutes an electric furnace. If the heat utilized is that which is developed in R, we necessarily have a type of furnace which makes use of electrodes, but if the heat utilized is only that which is developed in S we shall have a type

of furnace which is heated inductively and employs no electrodes. In all that follows we shall only give consideration to methods of developing useful heat inductively without iron, and without the use of electrodes.

If we define the closeness of coupling between circuits *I* and *II* to be the proportionate number of lines of magnetic induction developed by circuit *I* which interlink with circuit *II*, it is evident that this closeness of coupling may vary between unity and zero according as circuits *I* and *II* are intimately associated or are far separated.

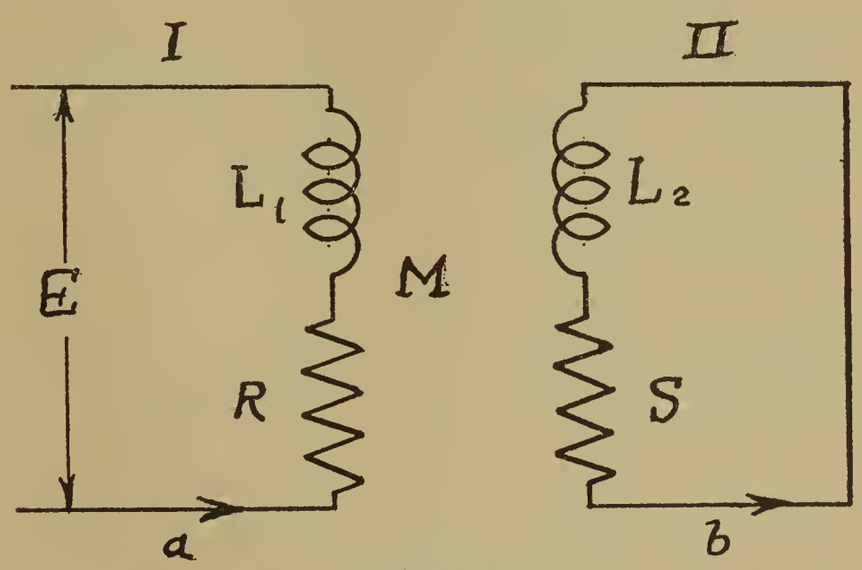


FIG. 1.

With a wide separation between circuits *I* and *II*, as between a sending and receiving circuit in radio practice, the use of iron to interlink the two circuits to increase the closeness of coupling is obviously impossible. But it is not essential, when the object to be attained is the sending of a signal from one circuit to the other, that any considerable proportion of the power supplied to circuit *I* should reach circuit *II*. If, on the other hand, the end sought is to transfer inductively a very considerable proportion of the power supplied to circuit *I* to circuit *II* for use in the production of mechanical effects or heat, then closeness of coupling between the primary and the secondary circuits is all essential. A closed iron magnetic circuit threading through the two electric circuits increases greatly the closeness of coupling between these, and therefore the use of iron has always been thought necessary and has heretofore been invariably employed.

As a first step toward ascertaining the theoretical possibilities of obtaining, without the use of iron, a rapid rate of transfer and a high efficiency in the transfer of energy between inductively coupled circuits, it will be highly useful to develop the mathematical relations which hold for a highly general case. To this we now proceed.

RATE OF HEATING AND THERMAL EFFICIENCY; GENERAL SOLUTION.

I. Rate of Heating in a Quaternary Circuit with E.M.F. Impressed on Primary Circuit.

Let the circuits here considered be represented in Fig. 2. The primary and secondary circuits which are inductively related constitute a pair *A*. The tertiary and quaternary circuits which

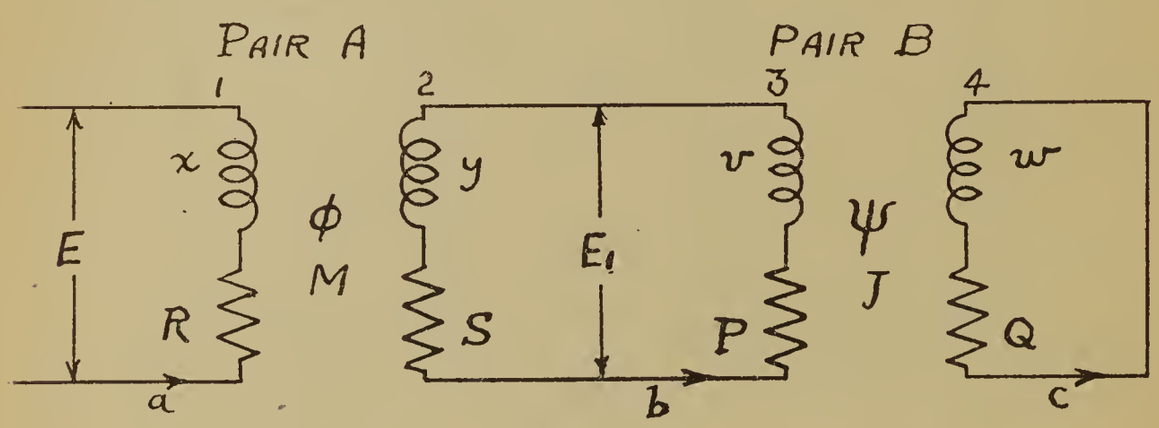


FIG. 2.

are inductively related constitute a pair *B*. It is assumed that because of their great separation there is no inductive relation between either of the members of the pair *A* with either of the members of the pair *B*.

Let x, y, v, w, be reactances.
Let R, S, P, Q, be resistances.
Let a, b, c, be currents.
Let M be the coefficient of mutual induction between *1* and *2*.
Let J be the coefficient of mutual induction between *3* and *4*.
Let φ be the degree of coupling of pair *A*.
Let ψ be the degree of coupling of pair *B*.
Let E be the e.m.f. of frequency N which is impressed on the primary circuit *1*.

It is first required to find an expression for the current C in circuit *4* and an expression for the rate of heating, $H = C^2 Q$, in circuit *4*, where Q is the ohmic resistance of the circuit and C^2 is the square of the absolute value of the current. We shall also require and will determine expressions for the current a in circuit *1* and for the current b in circuits *2* and *3* and we shall find an expression for the square of the absolute value of each.

The results sought are most readily obtained by making use of complex quantities.[3]

It is shown by Steinmetz ("Alternating Current Phenomena," Art. 31) that Kirchhoff's laws hold for alternating currents when expressed as follows:

(*a*) "The sum of all the e.m.f.'s acting in a closed circuit equals zero, if they are expressed by complex quantities, and if the resistance and reactance e.m.f.'s are also considered as counter e.m.f.'s."

(*b*) "The sum of all the currents flowing towards a distributing point is zero, if the currents are expressed as complex quantities." Thus we have (Fig. 2)

E = e.m.f. impressed on *1*.
Z_{12} = e.m.f. induced in *1* by unit current in *2*.
Z_{11} = counter e.m.f. in *1* due to unit current in *1*.
Then

$$E + Z_{11}a + Z_{12}b = O \qquad (1)$$

We have
Z_{21} = e.m.f. induced in *2* by unit current in *1*.
Z_{22} = counter e.m.f. in *2* due to unit current in *2*.
Z_{33} = counter e.m.f. in *3* due to unit current in *3*.
Z_{34} = e.m.f. in *3* due to unit current in *4*.
Then

$$Z_{21}a + (Z_{22} + Z_{33})b + Z_{34}c = O \qquad (2)$$

We have
Z_{43} = e.m.f. in *4* due to unit current in *3*.
Z_{44} = counter e.m.f. in *4* due to unit current in *4*.

[3] The writer here makes acknowledgment and expresses his thanks to Lieut. W. van B. Roberts for giving to the writer, at his request, valuable suggestions regarding the mathematical procedure employed.

Then

$$Z_{43}b + Z_{44}c = O \tag{3}$$

We know that

$Z_{12} = Z_{21} = Z_\alpha$ and $Z_{34} = Z_{43} = Z_\beta$ because the number of times circuit *1* is threaded by lines of force when unit current flows in *2* is the same as the number of times circuit *2* is threaded by lines of force when unit current flows in circuit *1*. The same considerations apply to circuits *3* and *4*.

Thus equations (*1*), (*2*), (*3*) become

$$E + Z_{11}a + Z_\alpha b = O \tag{4}$$

$$Z_\alpha a + (Z_{22} + Z_{33})b + Z_\beta c = O \tag{5}$$

$$Z_\beta b + Z_{44}c = O \tag{6}$$

Solving these three simultaneous equations we obtain,

$$a = \frac{Z_\beta{}^2 - (Z_{22} + Z_{33})\, Z_{44}}{D} E \tag{7}$$

$$b = \frac{Z_\alpha Z_{44}}{D} E \tag{8}$$

$$c = \frac{-Z_\alpha Z_\beta}{D} E \tag{9}$$

where $D = Z_{44}(Z_{11}Z_{22} - Z_\alpha^2) + Z_{11}(Z_{44}Z_{33} - Z_\beta^2)$.

The full complex expressions for the *Z's* are:

$Z_{11} = R + jx,$
$Z_{22} = S + jy,$
$Z_{33} = P + jv,$
$Z_{44} = Q + jw,$

The full complex expressions for Z_α and Z_β are: $Z_\alpha = R_\alpha + jx_\alpha$ and $Z_\beta = R_\beta + jx_\beta$.

We may however for physical reasons put $R_\alpha = O$. This is true because $Z_\alpha = Z_{12} = Z_{21}$ is the e.m.f. induced in *1* by unit current in *2*, or it is the e.m.f. in *2* induced by unit current in *1*, and in both these cases the e.m.f. depends only on the coefficient of mutual induction and the frequency and not at all on the resistance. Hence $Z_\alpha = jx_\alpha$ and similarly $Z_\beta = jx_\beta$.

Remembering the physical character of x_α and x_β, we have $x_\alpha = 2\pi NM$, where N is the frequency and M the coefficient of mutual induction between *1* and *2*. Similarly $x_\beta = 2\pi NJ$, where J is the coefficient of mutual induction between *3* and *4*.

Call L_1, L_2, L_3, L_4 the coefficients of self induction of *1, 2, 3, 4,* respectively, and n_1, n_2, n_3, n_4, the number of turns in the windings of *1, 2, 3, 4,* respectively. We know by physical reasoning that $M \propto n_1 n_2$ and $J \propto n_3 n_4$. Also we know that $n_1 \propto \sqrt{L_1}$, $n_2 \propto \sqrt{L_2}$, $n_3 \propto \sqrt{L_3}$, and $n_4 \propto \sqrt{L_4}$. It follows that $M \propto \sqrt{L_1 L_2}$ and $J \propto \sqrt{L_3 L_4}$. If we call φ the constant of proportionality in the first case and ψ that in the second case, we have $M = \varphi\sqrt{L_1 L_2}$ and $J = \psi\sqrt{L_3 L_4}$. Thus $x_\alpha^2 = 4\pi^2 N^2 \varphi^2 L_1 L_2 = \varphi^2 xy$ and $x_\beta^2 = 4\pi^2 N^2 \psi^2 L_3 L_4 = \psi^2 vw$. φ is the degree of coupling between *1* and *2* and ψ that between *3* and *4*.

If the coupling is 100 percent, then all lines of induction which link a primary due to a current in it will also link its secondary. Hence in this case $L_1 = M$. Likewise, all lines which link a secondary due to a current in it will also link its primary. Hence in this case $L_2 = M$. We therefore have, with 100 percent coupling, $M = \varphi\sqrt{L_1 L_2} = \varphi\sqrt{M^2} = \varphi M$, or $\varphi = 1$. If $M = O$, (since L_1 and L_2 do not equal zero) $\varphi = O$. Thus φ the degree of coupling, and likewise ψ may vary from *1* to *0*. Neither will vary in general, however, as a linear function of any space variable. We may now finally write,

$Z_\alpha = j\varphi\sqrt{xy}$ and

$Z_\beta = j\psi\sqrt{vw}$.

Putting in Eqs. (*7*), (*8*), and (*9*) the above values for the *Z's* we obtain for the values of the three currents the complex expressions,

$$a = \frac{w\left[v(1 - \psi^2) + y\right] - Q(S + P) - j\left[Q(y + v) + w(S + P)\right]}{D_1} E \qquad (10)$$

$$b = \frac{-w\varphi\sqrt{xy} + jQ\varphi\sqrt{xy}}{D_1} E \qquad (11)$$

$$c = \frac{\varphi\psi\sqrt{xyvw}}{D_1} E \qquad (12)$$

where

$$D_1 = Q\left[RS - xy(1 - \varphi^2)\right] + R\left[PQ - vw(1 - \psi^2)\right] - w(Ry + Sx) - x(Pw + Qv) + j\Big(Q(Ry + Sx) + w\left[Rs - xy(1 - \varphi^2)\right] + R(Pw + Qv) + x\left[PQ - vw(1 - \psi^2)\right]\Big).$$

Call r the real part of D_1 and p the imaginary part. Then

$$c = \frac{\varphi\psi\sqrt{xyvw}}{r + jp} E \tag{13}$$

Since the absolute value of a ratio is the ratio of the absolute value of the numerator to the absolute value of the denominator, we obtain for the square of the absolute value of c

$$c^2 = \frac{\varphi^2\psi^2 xyvw}{r^2 + p^2} E^2 \tag{14}$$

We now have as the expression for the rate of heating in circuit *4*

$$H_4 = c^2 Q = \frac{\varphi^2\psi^2 Qxyvw}{r^2 + p^2} E^2 \tag{15}$$

The rate of heating in circuits *2* and *3* will be given by the expression

$$H_{2+3} = b^2 (S + P) \tag{16}$$

Obtaining from Eq. (*11*) the value of b^2 we find

$$H_{2+3} = \frac{xy\varphi^2(w^2 + Q^2)(S + P)}{r^2 + p^2} E^2 \tag{17}$$

If we call e the real part and f the imaginary part of the numerator of the expression for a, Eq. (*10*), we obtain for the rate of heating in circuit *1*, $H_1 = a^2R$, or,

$$H_1 = \frac{(e^2 + f^2)R}{r^2 + p^2} E^2 \tag{18}$$

II. Thermal Efficiency in a Quaternary Circuit with E.M.F. Impressed on Primary Circuit.

If we treat the system of circuits shown in Fig. 2 as an apparatus for producing, electrically, useful heat in some one or more of the resistances R, S, P, or Q, the thermal efficiency of this apparatus is evidently that part of the supplied power which becomes converted into heat in the resistance or resistances selected, divided by the entire power supplied to the apparatus.

If we assume, properly, that all of the power supplied becomes converted into heat within the apparatus itself, then the entire power supplied, which we shall call P_s will be,

$$P_s = a^2R + b^2 (S + P) + c^2Q.$$

Here it must be remembered that a, b, c are absolute values of effective currents.

Thus the expression for the efficiency of electrical conversion into heat in the resistance Q or circuit *4* is

$$F_4 = \frac{c^2Q}{P_s} \tag{19}$$

and for the efficiency of electrical conversion into heat in the resistances S and P of circuits *2* and *3*, taken together, is,

$$F_{2+3} = \frac{b^2(S + P}{P_s} \tag{20}$$

and for the efficiency of conversion in the resistance R of circuit *1*, is,

$$F_1 = \frac{a^2R}{P_s} \tag{21}$$

Since we already know, from Eqs. (*15*), (*17*), and (*18*) the values of c^2Q, $b^2 (S + P)$ and a^2R, we know the value of P_s, which is their sum, and we can write down at once the values of F_4, F_{2+3}, and F_1.

These values are

$$F_4 = \frac{\varphi^2\psi^2xyvw}{D_2} Q \tag{22}$$

$$F_{2+3} = \frac{\varphi^2 xy(w^2 + Q^2)}{D_2}(S + P) \tag{23}$$

$$F_1 = \frac{e^2 + f^2}{D_2} R \tag{24}$$

$$\text{Now } D_2 = P_s \frac{r^2 + p^2}{E^2} \text{ or,}$$

remembering that, from Eq. (*10*),

$$e^2 = \Big(w\big[v(1 - \psi^2) + y\big] - Q(S + P)\Big)^2 \text{ and}$$

$$f^2 = \big[Q(y + v) + w(S + P)\big]^2 \text{ we have,}$$

$$D_2 = (e^2 + f^2)R + \varphi^2 xy\big[(w^2 + Q^2)(S + P) + \psi^2 vwQ\big].$$

We are now in possession, in Eqs. (*15*), (*17*) and (*18*), for the rate of heating in different parts of two pairs of coupled circuits (Fig. 2) and in Eqs. (*22*), (*23*), (*24*) for the efficiency of heating, of general expressions which will permit us to make complete analyses of our problem when studied from various points of view.

III. General Equations for Rate of Heating and Efficiency Applied to a Single Pair of Coupled Circuits.

If in Eq. (*15*) for the rate of heating in circuit *4* and Eq. (*22*) for the thermal efficiency we place $R = S = O$, $x = y$, and $\varphi = 1$, the e.m.f. E_1, applied at the terminals of *3* will be the same as the e.m.f. E applied at the terminals of *1*. The two new equations will then apply to the two coupled circuits of pair *B*, Fig. 2.

Again, if in Eq. (*17*) for rate of heating in circuits *2* and *3* and in Eq. (*23*) for thermal efficiency we place $P = v = \psi = O$ we shall obtain equations which apply to the two coupled circuits of pair *A*. This second set of equations should be identical in form with the first set. They will apply directly to the much used circuits of the single pair, as shown in Fig. 3.

Following the above procedure we find on putting $R = S = O$, $x = y$, and $\varphi = 1$, that Eq. (*15*) becomes,

$$H_4 = \frac{\psi^2 x^2 vwQ}{r^2 + p^2} E^2$$

and that r^2 and p^2 assume the values $r^2 = x^2\ (Pw + Qv)^2$ and $p^2 = x^2\ [PQ - vw\ (1 - \psi^2)]^2$.
Dropping the subscript from H_4 we now have,

$$H = \frac{\psi^2 vwQ}{(Pw + Qv)^2 + \left[PQ - vw(1 - \psi^2)\right]^2} E^2 \qquad (25)$$

Or expanding the denominator of Eq. (*25*) we obtain,

$$H = \frac{\psi^2 vwQE^2}{Q^2(P^2 + v^2) + 2PQvw\psi^2 + v^2w^2(1 - \psi^2)^2 + P^2w^2} \qquad (26)$$

Eq. (*26*) gives the rate of heating in the secondary circuit, as circuit *4* of Fig. 2, when the e.m.f. $E_1 = E$ is applied at the terminals of the primary circuit, as circuit *3* of Fig. 2.

In Eq. (*22*) for the efficiency we find

$$D_2 = x^2 \left[(w^2 + Q^2)\ P + \psi^2\ vwQ\right].$$

This equation now becomes, dropping the subscript of F_4,

$$F = \frac{\psi^2 vwQ}{P(w^2 + Q^2) + \psi^2 vwQ} \qquad (27)$$

Eq. (*27*) gives the thermal efficiency for heat developed in the secondary circuit, as circuit *4* of Fig. 2, when the e.m.f. $E_1 = E$ is applied at the terminals of the primary circuit, as circuit *3* of Fig. 2.
Again we find, on putting $P = v = \psi = O$ that Eq. (*17*), for rate of heating in circuits *2* and *3*, assumes the form

$$H = \frac{xy\varphi^2(w^2 + Q^2)SE^2}{r^2 + p^2} \qquad (28)$$

where r^2 and p^2 have the values,

$$r^2 = \left(Q\left[RS - xy(1 - \varphi^2)\right] - w(Ry + Sx)\right)^2 \text{ and}$$

$$p^2 = \left(Q(Ry + Sx) + w\left[RS - xy(1 - \varphi^2)\right]\right).$$

When the indicated operations are performed Eq. (*28*) reduces to

$$H = \frac{\varphi^2 xySE^2}{\left[RS - xy(1 - \varphi^2)\right]^2 + (Ry + Sx)^2} \qquad (29)$$

or in expanding the denominator, it assumes the final form,

$$H = \frac{\varphi^2 xySE^2}{S^2(R^2 + x^2) + 2\varphi^2 RSxy + x^2y^2(1 - \varphi^2)^2 + R^2y^2} \qquad (30)$$

It should be noted that Eq. (*30*) is identical in form with Eq. (*26*). It applies to the rate of heating in the secondary of a single pair of circuits as shown in Fig. 3.

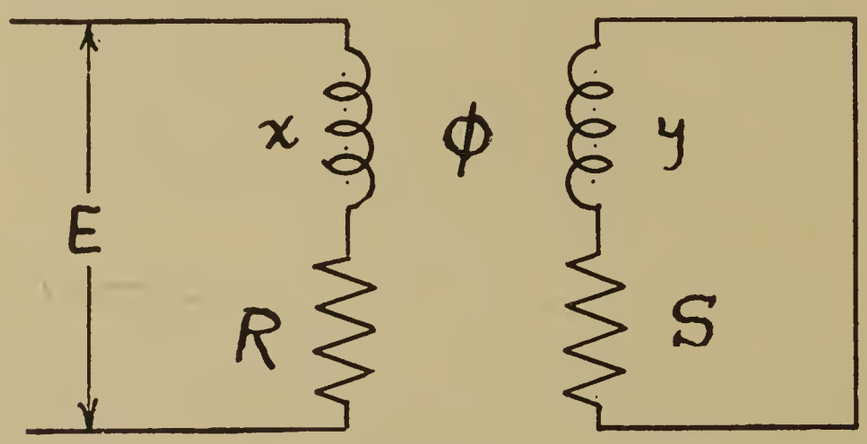


FIG. 3.

Putting $P = v = \psi = O$ in Eq. (*23*), for thermal efficiency, we find, dropping the subscript of F_{2+3}.

$$F = \frac{\varphi^2 xy(w^2 + Q^2)S}{R\left[(wy - QS)^2 + (Qy + wS)^2\right] + \varphi^2 xyS(w^2 + Q^2)}$$

which reduces to

$$F = \frac{\varphi^2 xyS}{R(y^2 + S^2) + \varphi^2 xyS} \qquad (31)$$

This is identical in form with Eq. (*27*). It applies to the thermal efficiency for heat developed in the secondary of a single pair of circuits as shown in Fig. 3.

Equations (*30*) and (*31*) contain the whole theory for the rate of heating and for the efficiency of transformation of applied power into heat in a secondary circuit when this contains no capacity and has a degree of coupling φ with a primary circuit upon which an alternating e.m.f. is impressed.

CONTROL OF CURRENT-PHASE IN SUPPLY-CIRCUIT; GENERAL EQUATIONS FOR A SINGLE PAIR OF COUPLED CIRCUITS.

In a pair of coupled circuits *I* and *II*, Fig. 4, the current *a* will not be in general in phase with the impressed e.m.f. *E*. To adjust the phase relations of *a* and *E*, a negative reactance x_c may be connected in shunt to the primary. Using a static condenser to secure this negative reactance we shall seek the expressions by which we may determine:

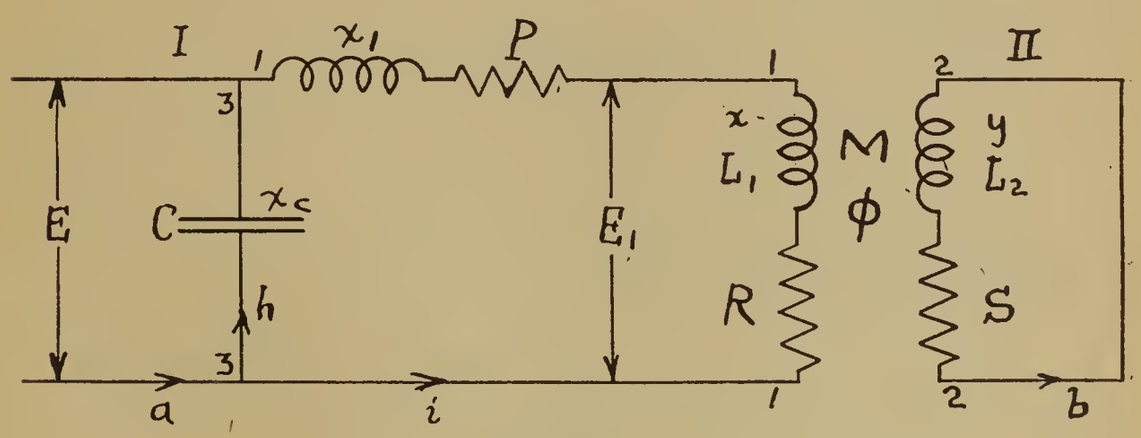


FIG. 4.

1—The complex expressions for the currents *a, h, i, b.*
2—The square of the absolute value of the currents *a, h, i, b.*
3—The phase, tan δ, of the current *a.*
4—The value of the reactance x_c to bring the current *a* into phase with *E*, also the value of the capacity *C*.
5—The ratio of the currents *a* and *i.*
6—The fall of potential $E - E_1$.

Let *E* be an e.m.f. of frequency *N* impressed on *I*.
Let x_1, x, x_c, y be reactances.
Let $x' = x_1 + x$.
Let *a, i, h, b* be currents.
Let *P, R, S* be resistances.
Let $r = P + R$.
Let L_1 be the self induction of branch circuit *1*.
Let L_2 be the self induction of secondary circuit *2*.
Let *C* be the capacity in the branch circuit *3*.
Let φ be the degree of coupling between *1* and *2*.
Let *M* be the mutual induction between *1* and *2*.
Let Z_{11} be the impedance of branch circuit *1*.

Let Z_{33} be the impedance of branch circuit *3*.
Let Z_{22} be the impedance of secondary circuit *2*.
Let $Z = \frac{Z_{11}Z_{33}}{Z_{11} + Z_{33}}$ be the impedance of the primary, *I*.
Let Z_{12} = e.m.f. induced in *1* by unit current in *2*.
Let Z_{21} = e.m.f. induced in *2* by unit current in *1*.
As previously shown,

$$Z_{12} = Z_{21} = Z_a = j\varphi\sqrt{xy}.$$

We now have,

$$E + aZ + bZ_{12} = O \tag{32}$$

and

$$iZ_{21} + bZ_{22} = O \tag{33}$$

By the law of branched circuits,

$$i = \frac{Z_{33}}{Z_{11} + Z_{33}}\, a = \frac{Za}{Z_{11}} \text{ and Eq. (33) becomes,}$$

$$a\, ZZ_{21} + bZ_{11}\, Z_{22} = O \tag{34}$$

From Eqs. (*32*) and (*34*) we find,

$$a = \frac{-EZ_{22}(Z_{33} + Z_{11})}{Z_{33}(Z_{11}Z_{22} - Z_a^2)} \tag{35}$$

$$b = \frac{EZ_a}{Z_{11}Z_{22} - Z_a^2} \tag{36}$$

and

$$i = \frac{Z_{33}}{Z_{11} + Z_{33}}\, a \tag{37}$$

and we have

$$h = \frac{E}{-jx_c}\,. \tag{38}$$

We have in connection with Fig. 4 the following expressions,

$$Z_{11} = (P + R) + j\,(x_1 + x),$$

and by writing $r = P + R$ and $x' = x_1 + x$,

$$Z_{11} = r + jx'.$$
$$Z_{22} = S + jy.$$
$$Z_{33} = -jx_c.$$
$$Z_a = j\varphi\sqrt{xy}.$$

$$Z = \frac{Z_{11}Z_{33}}{Z_{11} + Z_{33}} = \frac{-(r + jx')jx_c}{r + j(x' - x_c)} \text{ or}$$

$$Z = x_c\left(\frac{rx_c}{r^2 + (x' - x_c)^2} - j\,\frac{r^2 + x'(x' - x_c)}{r^2 + (x' - x_c)^2}\right) \qquad (39)$$

Z is the impedance of the primary I.
From Eq. (*35*)

$$a = E\,\frac{ry + S(x' - x_c) - j\left[rS - y(x' - x_c)\right]}{x_c\left[rS - x'y + \varphi^2 xy + j(ry + x'S)\right]} \qquad (40)$$

From Eq. (*36*)

$$b = \frac{E\varphi\sqrt{xy}}{(ry + Sx') - j(rS - x'y + \varphi^2 xy)} \qquad (41)$$

From Eq. (*37*)

$$i = \frac{-x_c}{x' - x_c - jr}\,a. \qquad (42)$$

From Eq. (*38*)

$$h = \frac{E}{-jx_c} = \frac{jE}{x_c}. \qquad (43)$$

From Eq. (*40*) we find the square of the absolute value of a to be,

$$a^2 = E^2\,\frac{(S^2 + y^2)\left[r^2 + (x' - x_c)^2\right]}{x_c^2\Big((S^2 + y^2)(r^2 + x'^2) - xy\varphi^2\left[y(2x' - x\varphi^2) - 2rS\right]\Big)} \qquad (44)$$

From Eq. (*44*) we have,

if $x_c = \infty$, $i = a$. And, if $r = R$ and $x' = x$,

$$i^2 = \frac{E^2(S^2 + y^2)}{(S^2 + y^2)(R^2 + x^2) - xy\varphi^2\left[xy(2 - \varphi^2) - 2RS\right]} \qquad (45)$$

Call the denominator $= D^2$, then

$$i^2 = \frac{E^2(S^2 + y^2)}{D^2}$$

By Eq. (*47*), p. 86,

$$b^2 = \frac{E^2 \varphi^2 xy}{D^2} .$$

Hence

$$b = i \frac{\varphi\sqrt{xy}}{\sqrt{S^2 + y^2}} \tag{46}$$

If we call $\varphi = 1$, and $x = y$, and $S = O$, $b = i$, which is the maximum possible value of b.

From Eq. (*41*) we obtain for the square of the absolute value of b,

$$b^2 = \frac{E^2\varphi^2 xy}{(S^2 + y^2)(r^2 + x'^2) - xy\varphi^2\left[y(2x' - \varphi^2 x) - 2rS\right]} \tag{47}$$

From Eqs. (*44*) and (*47*)

$$\frac{b^2}{a^2} = \frac{\varphi^2 xyx_c^2}{(S^2 + y^2)\left[r^2 + (x' - x_c)^2\right]} , \tag{48}$$

or

$$\frac{b}{a} = \frac{\varphi x_c\sqrt{xy}}{\sqrt{(S^2 + y^2)\left[r^2 + (x' - x_c)^2\right]}} . \tag{49}$$

From Eq. (*42*) we obtain for the square of the absolute value of i,

$$i^2 = \frac{x_c^2}{(x' - x_c)^2 + r^2} a^2 \tag{50}$$

or

$$i = \frac{x_c}{\sqrt{(x' - x_c)^2 + r^2}} a \tag{51}$$

If we write Eq. (*50*) in the form

$$i^2 = \frac{1}{\dfrac{x'^2}{x_c^2} - \dfrac{2x'}{x_c} + 1 + \dfrac{r^2}{x_c}} a^2,$$

we note that when $x_c =$ infinity, $i^2 = a^2$ or $i = a$.

From Eq. (*43*) we obtain for the square of the absolute value of h,

$$h^2 = \frac{E^2}{x_c^2} \tag{52}$$

or, we have

$$h = \frac{E}{x_c} = E \cdot 2\pi NC \tag{53}$$

We may now determine the phase of a using Eq. (*40*). To do this, and shorten the work, write,

$$\begin{aligned} a' &= ry + Sx' - Sx_c \\ b &= rS - yx' + yx_c \\ c &= rS - x'y + \varphi^2 xy \\ d &= ry + x'S. \end{aligned}$$

Eq. (*40*) now takes the form

$$a = \frac{E}{x_c} \cdot \frac{a' - jb}{c + jd} = \frac{E}{x_c} \cdot \frac{(a' - jb)(c - jd)}{c^2 + d^2}$$

or

$$a = \frac{E}{x_c} \left(\frac{a'c - bd}{c^2 + d^2} - j \frac{bc + a'd}{c^2 + d^2} \right) \tag{54}$$

The phase of a is then,

$$Tan\ \delta = - \frac{bc + a'd}{a'c - bd} \tag{55}$$

We obtain for the numerator of Eq. (*55*)

$$bc + a'd = (S^2 + y^2)\left[x'(x' - x_c) + r^2\right] - \varphi^2 xy \left[y(x' - x_c) - rS\right]$$

and for the denominator

$$a'c - bd = - rx_c(S^2 + y^2) + \varphi^2 xy\left[ry + S(x' - x_c)\right].$$

Thus we have as the full expression of Eq. (*55*)

$$Tan\ \delta = \frac{(S^2 + y^2)\left[x'(x' - x_c) + r^2\right] - \varphi^2 xy \left[y(x' - x_c) - rS\right]}{rx_c(S^2 + y^2) - \varphi^2 xy\left[ry + S(x' - x_c)\right]} \tag{56}$$

Eq. (*56*) can be written also in the form,

$$Tan\ \delta = \frac{r^2(S^2 + y^2) + \varphi^2 xyrS + (x' - x_c)\left[x'S^2 + y^2(x' - x\varphi^2)\right]}{rx_c(S^2 + y^2) - \varphi^2 xy\left[yr + S(x' - x_c)\right]}. \qquad (57)$$

Eq. (*56*) and also Eq. (*57*) reduce, when $\varphi = O$, to the simple form,

$$Tan\ \delta' = \frac{r^2 + (x' - x_c)x'}{rx_c} \qquad (58)$$

If we write Eq. (*58*) in the form

$$Tan\ \delta' = \frac{\frac{r^2}{x_c} + \frac{x'^2}{x_c} - x'}{r}$$ we see that when $x_c =$ infinity,

$$Tan\ \delta'' = \frac{-x'}{r} \qquad (59)$$

This equation shows, that when we use no condenser the current lags behind its e.m.f. by the angle

$$\delta'' = -\ tan^{-1}\ \frac{x'}{r}$$

The condition that the current a shall be in phase with the impressed e.m.f. E is that $Tan\ \delta = O$.
Thus placing the numerator of Eq. (*56*) or Eq. (*57*) equal to zero and solving for x_c we find,

$$x_c = x' + \frac{r^2(S^2 + y^2) + \varphi^2 xyrS}{S^2x' + y^2(x' - \varphi^2 x)} \qquad (60)$$

If in Eq. (*60*) $\varphi = O$

$$x'_c = x' + \frac{r^2}{x'} \qquad (61)$$

If in Eq. (*60*) $\varphi = 1$

$$x''_c = x' + \frac{r^2(S^2 + y^2) + xyrS}{S^2x' + y^2(x' - x)} \qquad (62)$$

The fall of potential over the impedance $Z = \sqrt{P^2 + x_1^2}$ is simply

$$E_{11} = E - E_1 = i\sqrt{P^2 + x_1^2} \qquad (63)$$

We may obtain the value of i from Eqs. (*51*) and (*44*).

CONDITIONS CONSIDERED GENERALLY FOR GREATEST RATE OF HEATING AND FOR MAXIMUM EFFICIENCY IN HEATING IN A SINGLE PAIR OF COUPLED CIRCUITS WITHOUT NEGATIVE REACTANCE.

In inductive heating with high-frequency currents there are two important queries, which bear directly upon the practicability of the methods and system here considered:

First: what theoretical limits determine the maximum rate of heating in the resistor of a secondary circuit when an e.m.f. is impressed on its primary?

Second: what is the theoretical maximum thermal efficiency of inductive apparatus when the apparatus is treated as a converter of electric energy supplied to its primary, into heat energy localized in the resistance of its secondary?

In answering these inquiries we may proceed as follows:

I. Independent Variables of Eqs. (30) and (31) and Their Partial Derivatives.

We may change independently in Eqs. (*30*) and (*31*) without affecting any other quantities, any of the five variables, φ, x, y, R, S. We may thus obtain from Eq. (*30*) the partial derivatives:

$$\frac{\partial H}{\partial \varphi}, \frac{\partial H}{\partial x}, \frac{\partial H}{\partial y}, \frac{\partial H}{\partial R}, \frac{\partial H}{\partial S}$$

and from Eq. (*31*) the partial derivatives:

$$\frac{\partial F}{\partial \varphi}, \frac{\partial F}{\partial x}, \frac{\partial F}{\partial y}, \frac{\partial F}{\partial R}, \frac{\partial F}{\partial S}.$$

When these partial derivatives are placed simultaneously equal to zero we shall be able to find one set of relations between certain of the quantities such that any departure from these relations

will diminish the rate of heating, and we shall be able to find another set of relations between certain of the quantities such that any departure will diminish the efficiency. Logically this does not mean that other relations may not exist which will give a greater rate of heating or a greater efficiency, but physical considerations enable us to assert that the relations found in the above manner are in reality the only maxima.

To shorten the expressions, call D the denominator of the quantity on the right of Eq. (*30*) and G the denominator of the quantity on the right of Eq. (*31*). The partial derivatives then become,

$$\frac{\partial H}{\partial \varphi} = 2xySE^2 \frac{S^2(R^2 + x^2) + y^2\left[R^2 + x^2(1 - \varphi^4)\right]}{D^2} \varphi \qquad (63\text{-}a)$$

$$\frac{\partial H}{\partial x} = \varphi^2 ySE^2 \frac{R^2(S^2 + y^2) - x^2\left[S^2 + y^2(1 - \varphi^2)^2\right]}{D^2} \qquad (63\text{-}b)$$

$$\frac{\partial H}{\partial y} = \varphi^2 xSE^2 \frac{S^2(R^2 + x^2) - y^2\left[R^2 + x^2(1 - \varphi^2)^2\right]}{D^2} \qquad (63\text{-}c)$$

$$\frac{\partial H}{\partial R} = -\varphi^2 xySE^2 \frac{2\left[R(S^2 + y^2) + S\varphi^2 xy\right]}{D^2} \qquad (63\text{-}d)$$

$$\frac{\partial H}{\partial S} = \varphi^2 xyE^2 \frac{y^2\left[R^2 + x^2(1 - \varphi^2)^2\right] - S^2(R^2 + x^2)}{D^2} \qquad (63\text{-}e)$$

and

$$\frac{\partial F}{\partial \varphi} = \frac{2RSxy(S^2 + y^2)\varphi}{G^2} \qquad (63\text{-}f)$$

$$\frac{\partial F}{\partial x} = \frac{RS\varphi^2 y(S^2 + y^2)}{G^2} \qquad (63\text{-}g)$$

$$\frac{\partial F}{\partial y} = \frac{RS\varphi^2 x(S^2 - y^2)}{G^2} \qquad (63\text{-}h)$$

$$\frac{\partial F}{\partial R} = \frac{-S\varphi^2 xy(S^2 + y^2)}{G^2} \qquad (63\text{-}i)$$

$$\frac{\partial F}{\partial S} = \frac{-R\varphi^2 xy(S^2 - y^2)}{G^2} \qquad (63\text{-}j)$$

II. Conditions to Give Maximum Rate of Heating.

If we place each of the partial derivatives of H equal to zero and solve the resulting relations for φ^4, x^2, y^2, R and S^2, we obtain from $\frac{\partial H}{\partial \varphi} = O$,

$$\varphi^4 = \frac{(R^2 + x^2)(S^2 + y^2)}{y^2 x^2} \qquad (63\text{-}k)$$

from $\frac{\partial H}{\partial x} = O$,

$$x^2 = R^2 \frac{S^2 + y^2}{S^2 + y^2(1 - \varphi^2)^2} \qquad (63\text{-}l)$$

from $\frac{\partial H}{\partial y} = O$,

$$y^2 = S^2 \frac{R^2 + x^2}{R^2 + x^2(1 - \varphi^2)^2} \qquad (63\text{-}m)$$

from $\frac{\partial H}{\partial R} = O$,

$$R = \frac{-S\varphi^2 xy}{S^2 + y^2} \qquad (63\text{-}n)$$

and from $\frac{\partial H}{\partial S} = O$,

$$S^2 = \frac{y^2\left[R^2 + x^2(1 - \varphi^2)^2\right]}{R^2 + x^2} \qquad (63\text{-}o)$$

Now the value for φ^4 shows that for the rate of heating to be a maximum φ should be greater than unity. As this is impossible we should make φ as great as possible and then consider it constant.

The value for R which will make the rate of heating a maximum is a negative value. As this is impossible we should make R small as possible and then consider it constant. We thus have left for solution the three simultaneous equations.

$$\frac{\partial H}{\partial x} = \frac{\partial H}{\partial y} = \frac{\partial H}{\partial S} = O \qquad (63\text{-}p)$$

To solve these readily place in Eq. (*63-o*) the value of y^2 taken from Eq. (*63-m*) and we obtain $S^2 = S^2$. This result shows that y may assume any value and still permit of obtaining maximum rate of heating.
Then give to y some value as,

$$y = y_0 \tag{63-q}$$

Place this value y_0, and the value for x^2 taken from Eq. (*63-l*), in Eq. (*63-o*), and we derive,

$$\frac{S^2}{y_0^2} = \frac{1 + \dfrac{(S^2 + y_0^2)(1 - \varphi^2)^2}{S^2 + y_0^2(1 - \varphi^2)^2}}{1 + \dfrac{S^2 + y_0^2}{S^2 + y_0^2(1 - \varphi^2)^2}}$$

This reduces to

$$S = y_0\sqrt{1 - \varphi^2} \tag{63-r}$$

Now place this value for S and the value y_0, in Eq. (*63-l*), and we have,

$$\frac{x^2}{R^2} = \frac{y_0^2(1 - \varphi^2) + y_0^2}{y_0^2(1 - \varphi^2) + y_0^2(1 - \varphi^2)^2}$$, and this reduces to

$$\frac{x^2}{R^2} = \frac{2 - \varphi^2}{2 - 3\varphi^2 + \varphi^4} = \frac{1}{1 - \varphi^2}$$. Hence

$$x = \frac{R}{\sqrt{1 - \varphi^2}} . \tag{63-s}$$

We are now in possession, in Eqs. (*63-q*), (*63-r*), (*63-s*), of values for y, S, and x (the three quantities taken as variables) which when placed in Eq. (*30*) for rate of heating, will show the greatest rate of heating which it is physically possible to obtain. Thus placing in Eq. (*30*) y_0 for y, $y_0\sqrt{1 - \varphi^2}$ for S, and $\dfrac{R}{\sqrt{1 - \varphi^2}}$ for x we obtain,

$$H = \frac{\varphi^2 E^2}{4R} \tag{64}$$

Eq. (*64*) tells us that when the variables S and x are given the most favorable values to correspond with any selected value of y the rate of heating will increase with the square of the degree of coupling, and the square of the applied voltage, and that it will diminish directly as the resistance of the primary circuit increases.

III. Maximum Efficiency Possible Consistent with Maximum Rate of Heating.

If we place these same values for y, S, and x in Eq. (*31*) for the efficiency we find,

$$F = \frac{\varphi^2}{2} \tag{65}$$

Thus Eq. (*65*) shows that when all quantities have been chosen to give the maximum rate of heating the efficiency cannot exceed 50 per-cent. This conclusion is in accord with a conclusion generally reached in connection with electric power apparatus and other devices, *i.e.*, that when conditions are arranged for a piece of apparatus to do external work at a maximum possible rate, power is consumed within the apparatus itself which is just equal to the power delivered externally by the apparatus.

IV. Condition For Maximum Efficiency.

We may now consider the condition for maximum efficiency. We see, Eq. (*63-f*), that $\frac{\partial F}{\partial \varphi}$ is a positive quantity and hence should be large as possible; that is the coupling should be close as possible. We need not consider, therefore, φ as a variable.

We see likewise, Eq. (*63-g*), that $\frac{\partial F}{\partial X}$ is a positive quantity. Hence x should be large as possible and this quantity also may be considered constant.

We see, Eq. (*63-i*), that $\frac{\partial F}{\partial R}$ is a negative quantity. Therefore the smaller we make R the greater the efficiency. Hence we should make R small as possible and treat it as a constant.

By placing $\frac{\partial F}{\partial y} = \frac{\partial F}{\partial S} = O$ we find,

$$S = y \tag{66}$$

simultaneously satisfies both of these relations. Hence in the choice of the variables, the sole condition for obtaining maximum efficiency is to make the resistance of the secondary equal its reactance.

If we now place $S = y$ in Eq. (*31*), for the efficiency, we find,

$$F = \frac{\varphi^2 x}{2R + \varphi^2 x} = \frac{\varphi^2 \pi N L_1}{R + \varphi^2 \pi N L_1} \tag{67}$$

where L_1 is the inductance, R is the resistance of the primary circuit, N is the frequency and φ is the degree of coupling.

V. Maximum Rate of Heating Possible Consistent with Maximum Efficiency.

We may now find the maximum rate of heating possible when we choose $S = y$, Eq. (*66*), the condition for maximum efficiency, instead of $S = y_0\sqrt{1 - \varphi^2}$, Eq. (*63-r*), a condition for maximum rate of heating. To do this replace S by y in Eq. (*30*) and we obtain,

$$H = \frac{\varphi^2 x E^2}{x^2\left[1 + (1 - \varphi^2)^2\right] + 2R\varphi^2 x + 2R^2} \tag{68}$$

Considering x variable, we may now find the value which x should have to obtain the maximum possible heating consistent with maximum efficiency, by placing $\frac{dH}{dX} = O$ and solving for x.

Doing this we find

$$x = \frac{R\sqrt{2}}{\sqrt{1 + (1 - \varphi^2)^2}} \tag{69}$$

If this value of x be now placed in Eq. (*68*) we obtain the maximum heating possible which is at the same time consistent with maximum efficiency.

Thus we find,

$$H_f = \frac{\varphi^2 E^2}{2R\left[\varphi^2 + \sqrt{2}\sqrt{1 + (1 - \varphi^2)^2}\right]} \tag{70}$$

where H_f is the maximum heating possible with maximum efficiency. If in Eq. (*70*) we put $\varphi = 1$, namely, make the coupling 100 percent, we obtain,

$$H_f = \frac{\varphi^2 E^2}{R} 0.207 \tag{71}$$

The maximum possible heating is, by Eq. (*64*),

$$H = \frac{\varphi^2 E^2}{4R}, \text{ hence,}$$

$$H_f = H0.828 = \frac{\varphi^2 E^2}{4R} 0.828 \tag{72}$$

Eq. (*72*) shows that if we arrange conditions to give the maximum possible heating with maximum efficiency, and make $\varphi = 1$ we will have a heating only 17.2 percent less than the maximum heating which it is possible to obtain with any arrangement. The conclusion is that we should strive for maximum efficiency by making the secondary resistance equal to the secondary reactance and the coupling 100 percent if possible, and sacrifice a little of the very maximum rate of heating obtainable which could be gotten by making $S = y_0\sqrt{1 - \varphi^2}$ instead of $S = y$. Choosing other values of φ we can form the following table:

Let $\varphi^2 = 0.1$, then $H_f = 0.9996H$
$\varphi^2 = 0.5$, then $H_f = 0.9638H$
$\varphi^2 = 0.8$, then $H_f = 0.892\ H$
$\varphi^2 = 1.0$, then $H_f = 0.828\ H$

We thus see that for small values of φ^2, H_f very nearly equals H. That is with loose coupling, when the condition is filled for maximum possible efficiency the conditions are also very nearly filled for maximum rate of heating.

VI. Modifications Required in the Equations When the Resistances Are a Function of the Frequency.

In the mathematical treatment which proceeds we have made, implicitly, five important assumptions:
First, the inductances and resistances consist of practically linear conductors which are without appreciable cross-section.

Second, the frequency N of the impressed voltage E is constant and single valued.

Third, the circuits are free from all capacity, distributed or bulked, except where, as indicated in Fig. 4, bulked capacity is intentionally introduced.

Fourth, the frequency is not so great, (or leading-in conductors are arranged non-inductively) that appreciable power is lost into space by electro-magnetic radiation.

Fifth, that whenever a partial derivative in respect to a reactance is taken it is the inductive coefficient factor of the reactance and not the frequency factor which is taken as variable.

Now, none of the assumptions are strictly in accord with actual practice and yet they must be made in order to sufficiently simplify the statements of the problems to make possible their mathematical solutions.

It must be remembered, therefore, that because we have made these necessary assumptions, the solutions which have been obtained will serve more, the purpose of guiding in design by showing the directions in which the various quantities vary, than by enabling us to obtain precise numerical results.

We may make, however, the resulting equations more nearly expressive of actual conditions of practice by remembering that all ohmic resistances are greater when carrying high frequency current than when carrying direct current and then properly introduce this idea into the final equations.

Thus if we call R the ohmic resistance of a conductor when carrying current of frequency N and R_1 its resistance when carrying direct current, we shall have,

$$R = R_1 f(N),$$

where $f(N)$ is some function of the frequency. The $f(N)$ is never less than one and when N is small, say 25 cycles per sec., $f(N)$ is practically unity, but if N is high, say 150,000 cycles per sec., $f(N)$ may be several times unity.

The value of $f(N)$ for any given high value of N will depend upon many factors, such as:

(a)—The specific resistance of the material of the conductor.

(b)—The shape of the cross-section of the conductor.

(*c*)—The dimensions of the cross-section of the conductor.

(*d*)—The number of strands per unit cross-section of the conductor.

(*e*)—The ratio of the cross-section of solid metal in the cross-section of the conductor to the total cross-section of the conductor.

(*f*)—The type of stranding or braiding of the individual wires making up the conductor.

(*g*)—The type of stranding or braiding of groups of wires which go to make up the conductor.

(*h*)—The type of winding in respect to the twisting of the strands.

(*i*)—The type of winding in respect to inductive or non-inductive winding.

(*j*)—The type of winding in respect to the closeness of association of two or more turns of the winding.

(*k*)—The completeness of the insulation of individual wires which are stranded together to form the conductor.[4]

If for R we now write $R_1 f(N)$ in Eqs. (*15*), (*22*), (*30*), (*31*), (*64*) and (*67*), we shall have a set of equations which much more closely express the physical facts, when the currents have high frequency, than the equations do before this substitution is made.

The value of the function, $f(N)$, for any particular case may be often obtained by suitably arranged experiments. Its experimental determination for certain typical cases would have high value not alone in connection with high frequency methods of electric heating, but also in connection with radio practice.

The equation (*67*), which expresses the maximum theoretical thermal efficiency obtainable, is the one which possesses for us the most interest in connection with the problem of electric heating with high frequency currents. If in this equation we replace R by $R_1 f(N)$ and rearrange terms we obtain,

$$F' = \frac{1}{\frac{f(N)}{N} \cdot \frac{R_1}{\varphi^2 \pi L_1} + 1} \qquad (73)$$

[4] These matters have been examined experimentally, using oscillatory currents, by E. F. Northrup and R. G. Thompson. See "The Losses in Cables at High Frequencies," Jour. of the Franklin Institute, July, 1916.

Eq. (*73*) tells us that, when the values, R_1, L_1 and φ, are fixed, the efficiency will approach unity as long as N continues to increase more rapidly than $f(N)$; a conclusion which points to the importance of so designing the inductor coil that its resistance to high frequency A.C. is as nearly as possible equal to its resistance to D.C. As long as $f(N)$ remains nearly unity, the thermal efficiency will increase indefinitely as the frequency N is increased. It must be remembered, however, that Eq. (*73*) is only true upon the assumption made, that we maintain at all times, $S = y = 2\pi N L_2$. Thus when we assume that an increase is made in N we must assume, for Eq. (*73*) to remain true, that a proper increase is made also in S. This increase in S can always be made physically, which justifies the assumption. Thus we may increase S, without much affecting the inductance L_2: (*a*) by increasing the specific resistance of the material of the resistor S, or (*b*) by diminishing the cross-section of the current path of the resistor S. Moreover when N increases there will occur a certain automatic adjustment toward a proper value of S due to the phenomenon of skin effect. For, as the frequency N is increased, calling for a higher value of S, the currents induced in S tend more and more to flow in the superficial layers of this resistor, thereby flowing in a more restricted path. It is probable that L_2 will also change a certain amount but not nearly so much as S.

When in Eq. (*70*) we replace R by $R_1 f(N)$ we obtain,

$$H_f' = \frac{\varphi^2 E^2}{2R_1 f(N)\left[\varphi^2 + \sqrt{2}\sqrt{1 + (1 - \varphi^2)^2}\,\right]} \tag{74}$$

which becomes, when $\varphi = 1$

$$H_f'' = \frac{E^2}{4.828 R^1 f(N)} \tag{75}$$

These two equations express, under the conditions stated, the maximum possible rate of heating obtainable which is at the same time consistent with maximum efficiency. Here again we note the importance of making designs to maintain both R_1 and $f(N)$ small as possible.

POWER TRANSFORMATION IN CONCENTRIC CYLINDERS.

In Fig. 5, *I* represents in vertical section and *II* in horizontal section two concentric cylinders which have O — — O' as a common axis. These cylinders are assumed to be so long that considerations of e.m.f.'s currents, fluxes, and heating, made respect-

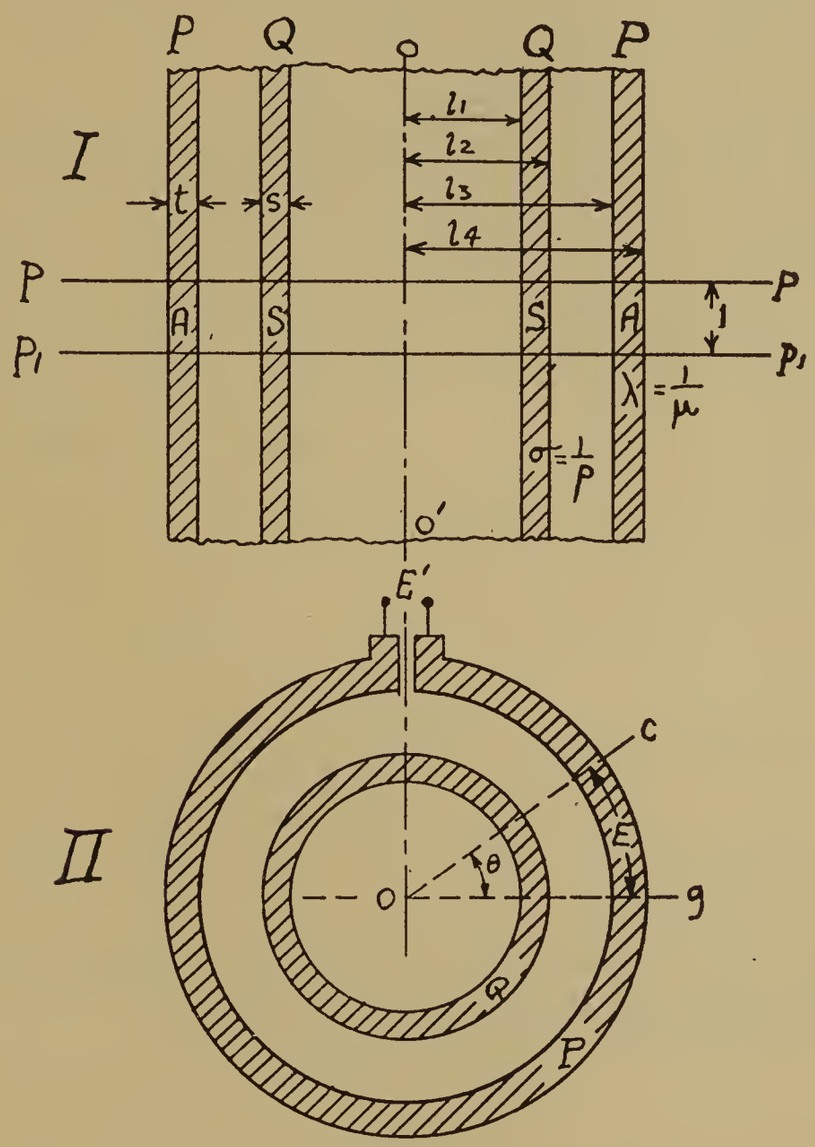


FIG. 5.

ing small portions of the cylinders included between two horizontal planes, as p — p and p_1 — p_1, will be true when "end effects" are ignored.

Let the electrical conductivity of the inner cylinder Q be σ and that of the outer cylinder P be λ.

Let the imaginary horizontal planes p — p and p_1 — p_1 be unit distance apart.

Call 1_1 and 1_2 the inner and outer radii of Q and 1_3 and 1_4 the inner and outer radii of P.

Assume that an electric current flows circumferentially in the outer cylinder P which results from an e.m.f. E' impressed at opposite sides of a vertical slot, as indicated in *II*, Fig. 5.

Assume that the cylinder Q forms a closed circuit and that any current in it flows circumferentially and results solely from an e.m.f. produced by induction.

We shall make the preliminary assumption that the impressed e.m.f. E' is of such low frequency, or that the cylinders have such thin walls, that the current distribution over the cross-sectional areas A and S (*I*, Fig. 5) may be treated as uniform.

Let $c - O - g$ (*II*, Fig. 5) be a unit angle = one radian.

Let i_0 be the absolute value of the current density in the area A of cylinder P.

Let i_1 be absolute value of the current density in the area S of cylinder Q.

Then the total absolute current in area A is

$$a = i_0 (1_4 - 1_3) \times 1 = i_0 t$$

where $t = 1_4 - 1_3$.

The total absolute current in area S is

$$b = i_1 (1_2 - 1_1) \times 1 = i_1 s$$

where $s = 1_2 - 1_1$.

The rate of heat generation in the area A throughout the length of the arc subtended by unit angle θ will be

$$W_0 = (i_0 t)^2 R_0$$

where R_0 is the ohmic resistance of the arc subtended by unit angle. The mean length of this arc is $\frac{1_3 + 1_4}{2}$. Hence $R_0 = \frac{1_3 + 1_4}{2t} \mu$ where $\mu = \frac{1}{\lambda}$ is the specific resistance of the material of the outer cylinder. Thus we obtain

$$W_0 = \frac{i_0^2 (1_3 + 1_4)}{2} t\mu \qquad (76)$$

or since $i_0^2 = \frac{a^2}{t^2}$

$$W_0 = \frac{a^2(l_3 + l_4)}{2t}\mu \tag{77}$$

The rate of heat generated in the area S throughout the arc subtended by unit angle θ will be

$$W_i = (i_1 s)^2 R_1$$

where R_1 is the ohmic resistance of the arc subtended by unit angle. The mean length of this arc is $\frac{l_1 + l_2}{2}$. Hence $R_1 = \frac{l_1 + l_2}{2s}\rho$ where $\rho = \frac{1}{\sigma}$ is the specific resistance of the material in the inner cylinder.

Thus we obtain

$$W_i = \frac{i_1^2(l_1 + l_2)}{2}s\rho \tag{78}$$

or since $i_1^2 = \frac{b^2}{s^2}$

$$W_i = \frac{b^2(l_1 + l_2)}{2s}\rho \tag{79}$$

If we call δ the ratio of the rate of heat developed in the inner cylinder to the rate of heat developed in the outer cylinder we obtain

$$\delta = \frac{W_i}{W_0} = \frac{i_1^2}{i_0^2} \cdot \frac{l_1 + l_2}{l_3 + l_4} \cdot \frac{s\rho}{t\mu} \tag{80}$$

or

$$\delta = \frac{b^2}{a^2} \cdot \frac{l_1 + l_2}{l_3 + l_4} \cdot \frac{t\rho}{s\mu} \tag{81}$$

These ratios hold equally whether a like angular portion or the whole of the circumference of each cylinder be considered.

We have next to find the absolute values of i^2_0 and i^2_1 or of a^2 and b^2 which placed in Eq. (*80*) or Eq. (*81*) will give us a complete solution for our problem upon the assumption made of uniform current distribution.

The value of the ratio $\frac{b^2}{a^2}$ may be obtained at once from information given in Eqs. (*20*), (*21*), (*23*), (*24*). By placing $P = v = \psi = O$ these equations apply to a single pair of coupled circuits where a is the absolute value of the current in the primary circuit produced by an e.m.f. impressed at its terminals and b is the absolute value of the current in the secondary circuit produced by induction.

From Eq. (*21*) $a^2 = \frac{P_s F_1}{R}$ and from Eq. (*20*) $b^2 = \frac{P_s F_{2+3}}{S + P}$.

Hence

$$\frac{b^2}{a^2} = \frac{R}{S + P} \cdot \frac{F_{2+3}}{F_1} \tag{82}$$

From Eq. (*23*) and Eq. (*24*) we obtain the ratio,

$$\frac{F_{2+3}}{F_1} = \frac{\varphi^2 xy(w^2 + Q^2)(S + P)}{(e^2 + f^2)R} \tag{83}$$

when $P = v = \psi = O$ (see page 78),

$$e^2 + f^2 = (w^2 + Q^2)\ (y^2 + S^2).$$

Thus Eq. (*83*) becomes, remembering that $P = O$,

$$\frac{F_{2+3}}{F_1} = \frac{\varphi^2 xyS}{(y^2 + S^2)R} \tag{84}$$

and this ratio when substituted in Eq. (*82*), and P in this equation is made equal to zero, gives,

$$\frac{b^2}{a^2} = \frac{\varphi^2 xy}{y^2 + S^2} \tag{85}$$

Also we have $D_i = 1_1 + 1_2$, the mean diameter of the inner cylinder and $D_0 = 1_3 + 1_4$, the mean diameter of the outer cylinder.

Substituting in Eq. (*81*) we obtain,

$$\delta = \frac{\varphi^2 xy}{y^2 + S^2} \cdot \frac{D_i}{D_0} \cdot \frac{t\rho}{s\mu} \tag{86}$$

In Eq. (*86*) if we take S as the resistance of the entire circumference of the inner cylinder included between the planes $p - p$

and $p_1 - p_1$ we must take y as the reactance of the same portion or zone of the inner cylinder and x as the reactance of the corresponding zone of the outer cylinder.

If we call R the resistance of the entire circumference of the zone of the outer cylinder we shall have,

$$R = \frac{\pi D_0}{t \times 1} \mu$$

and similarly

$$S = \frac{\pi D_i}{s \times 1} \rho$$

Thus

$$\frac{\rho}{\mu} = \frac{D_0 s S}{D_i t R}$$

which ratio placed in Eq. (*86*) gives

$$\delta = \frac{\varphi^2 x y}{y^2 + S^2} \cdot \frac{S}{R} \qquad (87)$$

We may vary the resistance S of the zone of the inner cylinder, without changing the dimensions of the cylinder, by varying the specific resistance of the material of which it is made. We are at liberty, therefore, to change S and to select that value for S which will make the ratio δ (rate of heat developed in a zone of inner cylinder to the rate of heat developed in a corresponding zone of outer cylinder) a maximum.

To find this value of S we take $\frac{d\delta}{dS}$ and equate it to zero.

Performing this operation on Eq. (*87*) we find $S = y$.

Good designing of any inductive heating apparatus would require that this condition has been met. We may therefore place $S = y$ in Eq. (*87*), and we obtain,

$$\delta_1 = \frac{\varphi^2 x}{2R} = \frac{\varphi^2 \pi N L_1}{R} \qquad (88)$$

In Eq. (*88*) we are to take L_1 as the self induction and R as the ohmic resistance of a zone of the outer cylinder, of any selected width, as unit width, measured in the direction of the

length of the cylinder and of any small thickness t such that the current distribution over the cross-section of the current-path may be considered uniform. Here N is the frequency of the e.m.f. which we assume is impressed at opposite sides of a vertical cut through the zone. (*E'*, Fig. 5, *II.*)

If the cylinder is very long we consider as a fair approximation that the distribution of the field of magnetic force is uniform over the cross-section of the outer cylinder. In this case

$$\varphi = \frac{D_i^2}{D_0^2}$$

That is, the proportionate number of the total number of lines of force developed which thread through the zone considered of the inner cylinder will be given by taking the ratio of the area of the inner cylinder to the area of the outer cylinder. Making, in Eq. (*88*), the substitution

$$\varphi^2 = \frac{D_i^4}{D_0^4}$$

we have

$$\delta_{11} = \frac{D_i^4}{D_0^4} \cdot \frac{\pi N L_1}{R} \qquad (89)$$

INFLUENCE OF SKIN EFFECT IN MODIFYING EQS. (*88*) AND (*89*).

In deducing Eqs. (*88*) and (*89*) it has been assumed that the current is uniformly distributed over the cross-section of the path in which it flows. In reality, however, the current distribution ceases to be uniform when the frequency is high.

The ratio of the current density at the inner circumference to the current density at the outer circumference of the outside cylinder continually increases with increasing frequency. Likewise the induced current flowing in the inside cylinder becomes distributed more in its outer circumferential region when the frequency is increased. In brief, it can be stated as a generalization that when one circuit, on which an e.m.f. is impressed, induces current in an adjacent circuit, the current tends to distribute over the cross-section of its path of flow in each circuit so as to

increase the closeness of coupling between the two circuits. This statement is capable of analytical proof, but it is more easily and vividly illustrated by the following simple experiment:

In Fig. 6 is shown an arrangement of circuits of low resistance wire. Current is produced by an e.m.f. *E*, at a frequency in the neighborhood of 200,000 cycles, which flows between points *1* and *2* by way of two concentric circular loops, *i* and *O*. Since the reactance of the inner and shorter path *i* is less than the reactance of the outer and longer path *O*, the inner path carries much the greater portion of the total current *I*. The ratio of the currents

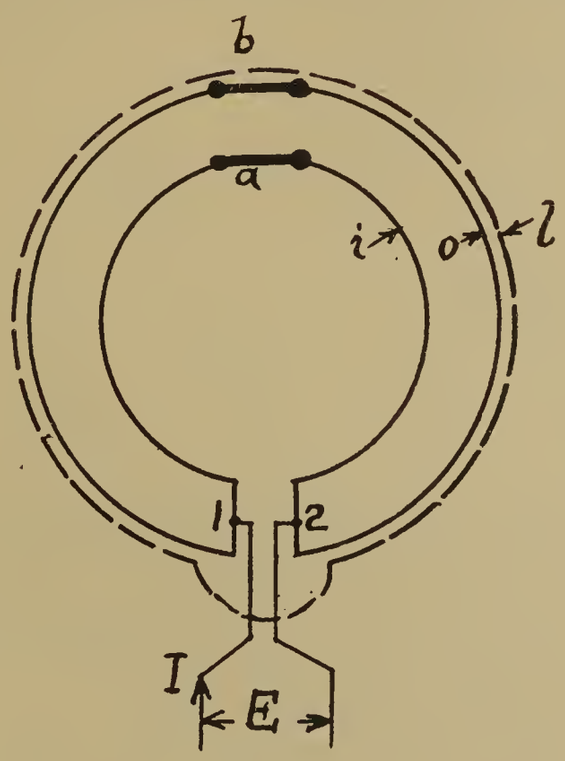


FIG. 6.

in paths *i* and *O* may be roughly observed by inserting a short length of high resistance wire, as *a* and *b*, in each path. These test wires will heat and glow. The test wire *a* in the inner path will glow much brighter than the test wire *b* in the outer path, showing that the former carries much the more current.

If, with conditions thus arranged, a closed loop *l* of wire of the same dimensions as the outer loop be now made to nearly coincide with the outer loop (as indicated in broken line in Fig. 6) the test wire *b* will glow more brightly than the test wire *a*. Thus the presence of the secondary loop *l* has the effect of causing a redistribution of the primary current in the two paths in such

manner as to increase the closeness of the coupling between primary and secondary circuits over what it would be if a redistribution of current were not effected.

Returning now to Eqs. (*88*) and (*89*) we may conclude from the above that when the frequency is high the effective ohmic resistance R of the outer cylinder is increased over its value for direct or slow frequency current. Calling then R the ohmic resistance of the conductor when carrying current of frequency N and R_1 its resistance when carrying direct current we should write $R = R_1F(N)$ where $F(N)$ is some function of the frequency, a quantity never less than one and possibly several times unity, when N is very large.

Eq. (*88*) now becomes,

$$\delta_1 = \frac{\varphi^2 \pi L_1}{R_1} \cdot \frac{N}{F(N)} \tag{90}$$

and Eq. (*89*) becomes

$$\delta_{11} = \frac{D_i^4}{D_0^4} \cdot \frac{\pi L_1}{R_1} \cdot \frac{N}{F(N)} \tag{91}$$

Of course the current also redistributes itself in the wall of the inner cylinder as well as in the wall of the outer cylinder, but we are assuming that the specific resistance of the material of the inner cylinder is always chosen, to maintain the relation $S = y$, whatever the frequency which we select.

Eqs. (*90*) and (*91*) give very important information on what is possible in the matter of obtaining a high rate of heat production in the walls of a conducting mass, where it is wanted, as compared with the rate of heat production in the inductor coil which surrounds the mass, where it is not wanted.

We conclude from the theory leading to Eqs. (*90*) and (*91*), which theory has been amply and strikingly verified by experiments, that the ratio δ_1 may be made several hundred fold.

It should be noted that as both L_1 and R_1 increase with the square of the number of primary turns which fill any selected winding channel as *A*, Fig. 5, that the value of δ_1 is unaffected by changing the number of turns, provided the impressed voltage always maintains the current density in the conductor the same. That is, we

must always keep the impressed voltage per turn the same to maintain the current density constant.

If in general we have the value of a ratio, as $\frac{u}{w}$, where u expresses power which is consumed usefully and w expresses power which is wasted, then $\frac{u}{w + u}$ becomes an expression for the efficiency of the apparatus employed for producing power transformation. In Eq. (*90*) we can consider $\delta_1 = \frac{u}{w}$ and hence the expression for the thermal efficiency of the heat-producing apparatus becomes,

$$F = \frac{u}{w + u} = \frac{\varphi^2 \pi L_1 N}{R_1 F(N) + \varphi^2 \pi L_1 N} \qquad (92)$$

or

$$F = \frac{1}{\frac{F(N)}{N} \cdot \frac{R_1}{\varphi^2 \pi L_1} + 1} \qquad (93)$$

If we compare Eq. (*93*) with Eq. (*73*) we note that the two equations are identical except that in Eq. (*93*) we have $F(N)$ instead of $f(N)$. Thus starting our line of reasoning from a particular case we have arrived at an expression [Eq. (*93*)] for the efficiency, of exactly the same kind as we reached [Eq. (*73*)] when starting from the perfectly general case.

It is proper to write $F(N)$ for the particular case instead of $f(N)$ used for the general case, because the manner in which the resistance will vary with the frequency, will depend to a large degree, upon the particular form which the primary inductor coil assumes. $f(N)$ is therefore to be taken as an expression more general in form than the expression $F(N)$.

HEAT PRODUCTION AND RETENTION.

If an elevation of temperature in a crucible and its contents is to be secured, the heat supplied to the crucible must be wholly or in part retained.

For retaining the heat developed in the crucible by induction,

either one or a combination of two distinct methods of heat insulation may be used.

By *Method I* we may heat-insulate by filling the space between the crucible (secondary circuit) and the inductor coil (primary circuit) with the best heat-insulator obtainable, and place no heat insulation on the outside of the inductor coil.

By *Method II* we may heat-insulate entirely on the outside of the inductor coil itself, which may then be supposed to fit quite closely round the crucible.

Method I necessitates a more or less loose coupling between the primary and secondary, but as the inductor coil itself is relatively cool it is easy to electrically insulate for a high e.m.f. at its terminals, and a high e.m.f. must be used to give the requisite current density in the primary conductor, if this consists of many turns.

In Method II the inductor coil, by reason of its close proximity to the heated crucible, will eventually attain the same temperature as the crucible. Hence this method limits the temperature attainable to the melting temperature of the material of the inductor coil itself. Furthermore, no materials are known which will electrically insulate for several hundred volts when the temperature begins to exceed 1,200° C. Hence when *Method II* is employed we become limited to the use of a few hundred volts at the terminals of the inductor coil.

It is concluded that *Method I* should be employed when very high temperatures are sought and *Method II* when medium temperatures are required, because in using *Method II* very close coupling and consequently good efficiencies may be expected.

TEMPERATURE EFFICIENCY.

In the heat treatment of material it is often a chief requirement to be able to obtain a high temperature for the purpose of studying the physical and chemical action of temperature upon a given mass of material. The quantity of the material to be subjected to the action of temperature may not be large. The desired ends are attained if a high temperature throughout a relatively small volume is secured. The cost of the supplied electric energy used is insignificant as compared with the importance of the results sought, and it follows that conditions should be

arranged to give the results without special regard to the cost of the kilowatt-hours needed.

On the other hand, the end sought may be the commercially important one of producing fusion (as in the melting of metals and glass) of large quantities of materials at the least possible cost.

In this case the highest temperature desired is only that which will melt and slightly superheat the product being treated. When the electric furnace is used for this purpose, it is highly important to arrange conditions for obtaining the best possible thermal efficiency, at the required temperature.

We shall now consider:

(*a*) How to obtain the highest temperature when *Method I* is used to give heat insulation.

(*b*) How to obtain the best thermal efficiency at or slightly above the fusion temperature of the material treated, when *Method II* is used to give heat insulation.

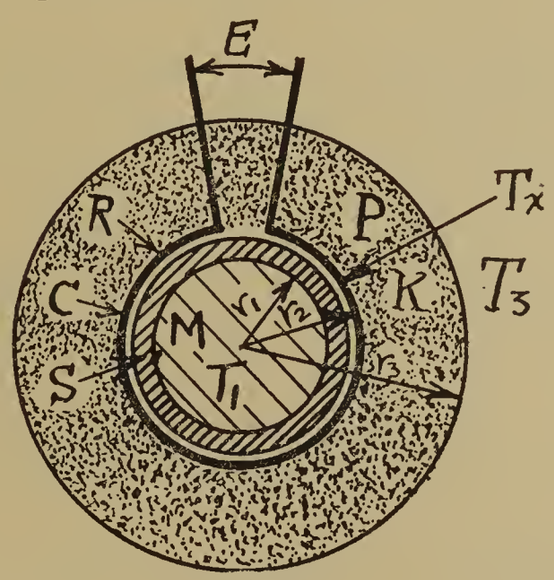


FIG. 7.

I. Condition for Highest Temperature Using Method I.

In Fig. 7, (*1*) is the crucible to be raised to a temperature T_2, (*3*) is the inductor coil which will attain a temperature T_1, and (*2*) is heat-insulating material of thermal conductivity k.

We shall confine attention to a zone of unit width taken near the central section, normal to the common axis of (*1*) and (*3*). The flow of heat from (*1*) when expressed in watts is,

$$W = S_f k \ (T_2 - T_1) \qquad (94)$$

Here S_f is a constant, the so-called "Shape-factor." If a is the radius of the outside wall of the crucible and b the radius of the inside surface of the inductor coil, the exact value of the shape-factor, for a zone of unit width is,

$$S_f = \frac{2\pi}{\log_\epsilon \frac{b}{a}}$$

For our purpose we may use the approximate formula

$$S_f = \frac{2\pi}{\frac{b - a}{a}} = \frac{2\pi a}{b - a}$$

Eq. (*94*) now becomes,

$$W = \frac{2\pi a}{b - a} kT \qquad (95)$$

where $T = T_2 - T_1$.

The input of heat by induction will depend upon how the values are chosen of S, the resistance taken circumferentially, of the zone of the inner cylinder, and x the reactance of the corresponding zone of the inductor coil. If conditions for maximum rate of heating have not been met then Eq. (*30*), page 80, applies.

If the two conditions for maximum rate of heating possible have been met, *i.e.*,

$$S = y_0\sqrt{1 - \varphi^2} \text{ and } x = \frac{R}{\sqrt{1 - \varphi^2}}$$

then Eq. (*64*), page 90, applies.

If the sole condition for maximum efficiency, *i.e.*, $S = y$, has been met, then Eq. (*68*), page 92, applies.

If the conditions for maximum rate of heating possible consistent with maximum efficiency have been met, *i.e.*,

$$S = y \text{ and } X = \frac{R\sqrt{2}}{\sqrt{1 + (1 - \varphi^2)^2}}$$

then Eq. (*70*), page 92, applies.

Temperature equilibrium will be finally established in each case when the rate of input and rate of outflow of heat are equal.

In seeking to obtain the highest possible temperature we would disregard thermal efficiency and arrange the values of S and x to give the greatest rate of heating possible. In this case by Eq. (*64*) we would have

$$H = \frac{\varphi^2 E^2}{4R}$$

Placing this value for H equal to the value for W [Eq. (*95*)] and solving for T we obtain,

$$T = \frac{(b - a)\varphi^2}{a} \cdot \frac{E^2}{8\pi Rk} \qquad (96)$$

We know that φ is dependent upon the radius a of the inner cylinder. As we wish to find the best value to give a when all other quantities are fixed (except x and S which are assumed to be always so chosen that Eq. (*70*) holds true) we must express φ in terms of a.

If the cylinder and inductor coil are very long and the zone considered is of small width we can with little error write

$$\varphi = \frac{\pi a^2}{\pi b^2} \text{ or } \varphi^2 = \frac{a^4}{b^4}$$

Placing this value for φ^2 in Eq. (*96*) we obtain,

$$T = (ba^3 - a^4)\frac{E^2}{8\pi Rkb^4} \qquad (97)$$

Thus we find for $\frac{\delta T}{\delta a} = O$, $3ba^2 - 4a^3 = O$, or

$$a = \tfrac{3}{4}b. \qquad (98)$$

This result shows that, very approximately, we should make the radius of the crucible to be heated three-fourths the radius of the inductor coil.

This result has been obtained by making the tacit assumption that the radial thickness of the inductor coil is negligible and that the current in the crucible wall flows close to its circumference.

II. Conditions for Producing Fusion at a Given Temperature, Using Method II.

In Fig. 8 is represented diagrammatically in cross-section a cylindrical induction furnace the inductor coil of which, following *Method II,* is heat-insulated externally.

Let r_1 = radius of the cylindrical mass, M, which we shall call the melt.

Let r_2 = radius of inductor coil C.

Let r_3 = radius of the cylindrically formed heat insulation.

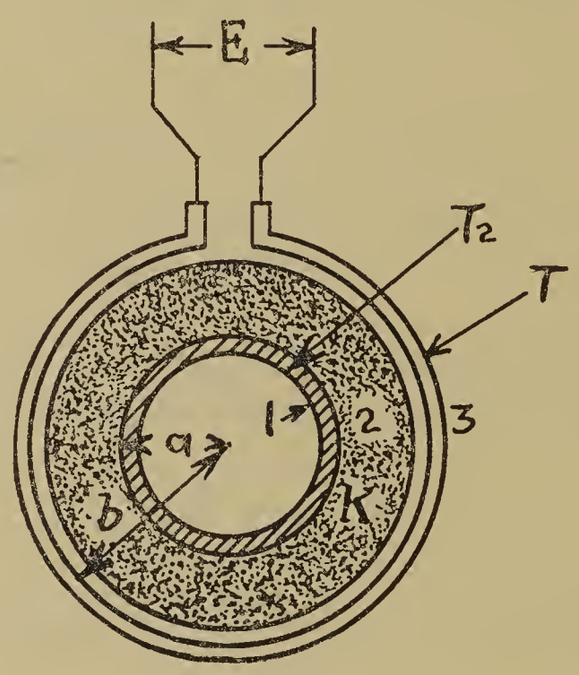


FIG. 8.

Assume that S, which may represent the wall of a non-electrically conducting crucible, has a thermal conductivity k. To simplify the analysis assume this is the same as the thermal conductivity of the heat insulating material P, external to the inductor coil, C. Assume that a current a at frequency N passes through the inductor coil C of ohmic resistance R and by its inductive effect produces heat in M. Assume further that the melt M has reached the temperature T_1, at which fusion begins. Then, until fusion is complete the temperature will remain constant at T_1 and all heat which enters and remains in M is absorbed as latent heat of fusion and produces the desired melting of the material. We shall define the melting efficiency of the furnace

as the ratio of the heat (expressed in watts) absorbed in the unit of time as latent heat of fusion, to the total watts supplied electrically at the furnace terminals (*1*), (*2*).

Two cases may be considered; the ohmic resistance R of the inductor coil may be so small that we can call it negligible, or it may be of such value that the a^2R heat generated in it will maintain its temperature exactly equal to the temperature T_1 of the melt while fusion is in progress. In this latter case, all heat which leaves the central region of the cylindrical furnace, flowing radially outward, will leave from the outer surface of the inductor coil. There will be no exchange of heat by conduction between the melt and the inductor coil because these are both at the same temperature T_1, the temperature of fusion.

Call F_m the melting efficiency when R is negligible. Then if we call W the heat generated by induction in the melt in unit time this will be the total heat supplied electrically. Since the melt is at a higher temperature than the region outside the heat insulation P, some of the heat generated in the melt will not disappear as latent heat of fusion but will flow radially outward by conduction.

If we call W_0 the heat lost in this manner, we have for the heat which remains in the melt to change the melt from the solid to the fluid state, the difference between the two quantities, W and W_0. Thus we have as the expression for the melting efficiency, when R is negligible,

$$F_m = \frac{W - W_0}{W} \qquad (99)$$

Let R not be negligible and so choose R that with the current A through the inductor coil necessary to put W_i watts into the melt, the A^2R loss in the inductor coil brings its temperature to the temperature of the melt. Then the total heat supplied electrically will be W_i, the heat which goes into the melt by induction, plus the A^2R heat developed in the inductor coil. Since the temperature of the inductor coil is equal, by arrangement of the resistance R, to the temperature of the melt, no part of W_i will be lost from the melt and all of it will be effective for producing fusion. Hence the melting efficiency in this case is given by the expression,

$$F_f = \frac{W_i}{W_i + A^2R} \tag{100}$$

The heat which will flow in the unit of time from the outer surface of the inductor coil, when this is at the temperature of the melt, will not in general be the same as W_0, that which flows from the surface of the melt when R is negligible, because the radius of the inductor coil is r_2, a quantity greater than the radius r_1 of the melt. We may, therefore, call W_c the rate of flow of heat from the inductor coil and we shall have $A^2R = W_c$. Since the temperatures T_1 of the melt and the inductor coil are given, also the temperature T_3 of the surroundings, also the quantities r_2, r_3 and k, we can determine the value of W_c and so find the value to assign to R to fill the required condition, that the melt and inductor coil shall be at the same temperature. The value of the current A, as well as its frequency N, are arbitrary, and A must be chosen according to the rapidity of fusion desired. We can now write Eq. (*100*) in the form

$$F_f = \frac{W_i}{W_i + W_c} \tag{101}$$

W_0 in Eq. (*99*) is perfectly determinable in the same way as is W_c. In analogy with the expression for heat flow given in Eq. (*95*) we have as approximate expressions for W_0 and W .

$$W_0 = \frac{2\pi r_1}{r_3 - r_1} kT \tag{102}$$

where $T = T_1 - T_3$ and

$$W_c = \frac{2\pi r_2}{r_3 - r_2} kT \tag{103}$$

Thus Eq. (*99*) becomes

$$F_m = \frac{W(r_3 - r_1) - 2\pi r_1 kT}{W(r_3 - r_1)} \tag{104}$$

and Eq. (*101*) becomes

$$F_f = \frac{W_i(r_3 - r_2)}{W_i(r_3 - r_2) + 2\pi r_2 kT} \tag{105}$$

This last equation is only true upon the assumption that we have made

$$R = \frac{W_c}{A^2} = \frac{2\pi r_2}{A^2(r_3 - r_2)} k(T_1 - T_3) \qquad (106)$$

We may make such an adjustment of the current A and the resistance R that the melt receives by induction and holds, as much heat as it does in the case where the resistance of the inductor is entirely negligible. Thus we shall have $W_i = W - W_0$, or

$$W_i = \frac{W(r_3 - r_1) - 2\pi r_1 kT}{r_3 - r_1}$$

If this value for W_i is substituted in Eq. (*105*) we obtain,

$$F_f' = \frac{\dfrac{W(r_3 - r_1) - 2\pi r_1 kT}{r_3 - r_1}(r_3 - r_2)}{\dfrac{W(r_3 - r_1) - 2\pi r_1 kT}{r_3 - r_1}(r_3 - r_2) + 2\pi r_2 kT} \qquad (107)$$

If we consider that the inductor coil fits closely round the melt, namely, $r_2 = r_1$, Eq. (*107*) reduces to

$$F_f'' = \frac{W(r_3 - r_1) - 2\pi r_1 kT}{W(r_3 - r_1)} \qquad (108)$$

which is identical with Eq. (*104*).

We thus arrive at the important conclusion: If the inductor coil fits closely round the melt and all the heat insulation is outside of the inductor coil we may, without loss of melting efficiency, give to the inductor coil such an ohmic resistance that the heat generated in it by A^2R heating will raise its temperature to be equal to that of the melt; for in this case the A^2R heat is just that which would flow outward by conduction from the melt at fusion temperature if there were no inductor coil.

For brass melting, where the temperature is maintained at about 1,100° C., a high melting alloy as nichrome or monel metal may therefore be used for the inductor coil in place of copper, which latter could not be used because it would be above its fusion temperature.

FIELD-ENERGY FORMULA FOR INDUCTIVE HEATING.

We shall develop a very simple formula which may be used to calculate to a fairly close approximation, the power consumed in heat by a cylindrical solid mass subjected to high frequency induction.

In Fig. 8a let M be a cylindrical mass of diameter d surrounded by a cylindrical coil of inside diameter D. It is assumed that the mass M has a conductivity suitable for inductive heating with high frequency currents. Thus its resistivity should lie within such

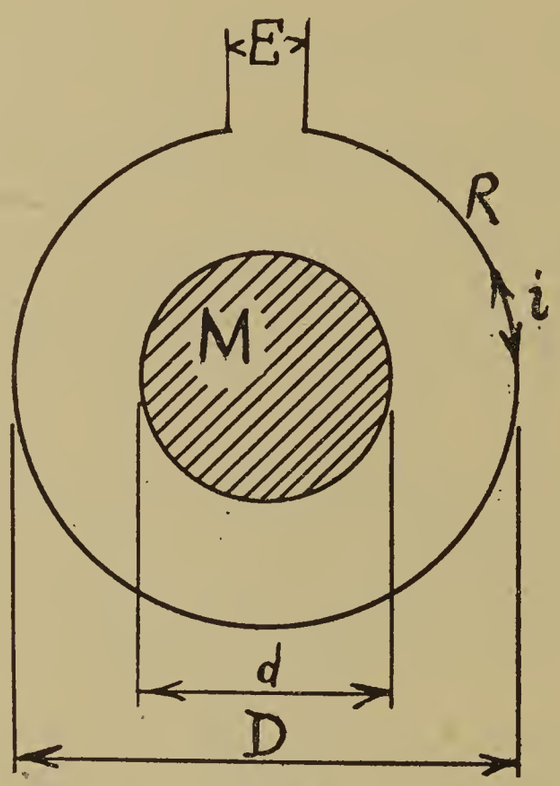


FIG. 8 a.

limits as 10 to 1,000 microhms per cm. cube. The axial length of the inductor coil and the mass M may have any value but are assumed to be equal. The inductor coil may have any number of single layer turns. We shall call its ohmic a.c. resistance R and its self induction L. When the coil is traversed by a current i, it is threaded by a magnetic flux. This flux is distributed with approximate uniformity over a plane inclosed by a turn of the coil. A certain proportion of this flux passes through a plane inclosed by the circumference of the mass M. From the standpoint of inductive action of the coil on the mass M, the degree of coupling between the coil and the mass is φ, where φ equals

the ratio of the lines of induction which thread the mass M to the total number of lines of induction associated with the coil.

The total number of lines of induction which are associated with the coil when carrying an instantaneous current i, represents the total electro-magnetic energy which is given to the electro-magnetic field when the current in the coil is raised from O to i. If the current attains a maximum value I, the numerical value of the energy given to the electro-magnetic field is

$$W = \tfrac{1}{2}LI^2 \qquad (108\text{-}a)$$

The energy given to that portion of the electro-magnetic field which is occupied by the mass M is

$$W_1 = \tfrac{1}{2}\,\varphi\, LI^2 \qquad (108\text{-}b)$$

If the current which flows through the coil is alternating energy W_1 is put into that portion of the field occupied by the mass M each time the current rises to a maximum, and, if the frequency of the current is N, this will occur $2N$ times in the unit of time.

If we assume that the energy W_1 becomes entirely changed into heat by virtue of the eddy currents which are caused to circulate in the mass M by the rapid entrance and exit of the lines of induction which thread the mass M, power in the form of heat will be supplied to the mass M, which is

$$P_1 = 2NW_1 = \varphi\, LNI^2 \qquad (108\text{-}c)$$

When eddy currents are induced in a continuous mass of material their reactive effect on the current in the inductor coil which gives rise to these eddy currents is practically *nil*. Thus if E is the effective value of the e.m.f. applied at the terminals of the inductor coil we have, to a very close approximation,

$$I^2 = 2i^2 = \frac{2E^2}{R^2 + (2\pi NL)^2} \qquad (108\text{-}d)$$

where i is the effective value of the high frequency alternating current through the inductor coil.

Substituting this value of I^2 in Eq. (*108-c*) we obtain as an expression for the power consumed as heat in the mass M,

$$P_1 = \frac{\varphi LN \cdot 2E^2}{R^2 + 4\pi^2N^2L^2} = \frac{\varphi E^2}{\dfrac{R^2}{2NL} + 2\pi^2NL} \qquad (108\text{-}e)$$

In actual cases where the frequency is high R^2 will be a small quantity as compared with $2NL$, and we may write

$$P = \frac{\varphi E^2}{2\pi^2 NL} \qquad (108\text{-}f)$$

If we write Eq. (*108-f*) in the form

$$P = \frac{\varphi}{\pi} \cdot E \frac{E}{2\pi NL}$$

and we remember that, when R is neglected,

$$i = \frac{E}{2\pi NL}$$

we obtain,

$$P = \frac{\varphi}{\pi} Ei \qquad (108\text{-}g)$$

Thus, the power consumed in a continuous cylindrical mass of conducting material, which is subjected to the inductive action of a cylindrical coaxial inductor coil of equal length with the mass, is the product of the volt-amperes multiplied by the constant $\frac{\varphi}{\pi}$.

This result presupposes that the frequency has an order of magnitude of 10^4 to 10^5 cycles per second and that the mass heated has a conductivity of the order of magnitude of mercury. If the coil and cylindrical mass are as long or longer than the diameter of the coil and the latter is about ⅔ to ⅞ of the diameter of the coil, then, very approximately,

$$\varphi = \frac{0.8d^2}{D^2}$$

and Eq. (*108-g*) becomes

$$P = \frac{0.8d^2}{\pi D^2} Ei\ . \qquad (108\text{-}h)$$

Since the current i carried by the inductor coil is readily measured and also the potential at its terminals (using an electrostatic voltmeter) Eqs. (*108-g*) and (*108-h*) become exceedingly useful for rapidly calculating the power being consumed as heat in the mass being heated.

If we do not know the current i, L can be calculated by the formula No. 74 for self induction given in the Bulletin of the Bureau of Standards, Vol. 8, No. 1, pp. 118 and 194.

PART II: APPLICATIONS OF THEORY.

PRODUCTION OF HIGH FREQUENCY CURRENTS.

There are three well known and much used methods for the production of high frequency currents (say, between the limits $N = 10^4$ and $N = 10^6$ cycles per second). These are:

(a)—The Arc Method of V. Poulsen: Here sustained oscillations are maintained in an oscillatory current circuit, which contains a special type of arc, by continuously receiving energy from a source which contains a direct e.m.f.[5]

(b)—The Tesla Oscillatory Current Circuit: In this method the oscillatory current circuit contains a discharge gap which breaks down and heals at more or less regular intervals. At each rupture of the gap the charged capacity discharges in an oscillatory manner, the oscillations damping out logarithmically. This type of circuit may be fed from a source of high-voltage direct e.m.f. but high-voltage alternating e.m.f. is more commonly employed.

(c)—The High Frequency Alternating Current Generator: With this method currents of a frequency as high as 2×10^5 cycles per second have been produced for radio work. The tendency, however, in radio practice is to employ lower frequencies, and a generator yielding 200 K.W. at 100 volts and giving a frequency of 3×10^4 is in service for wireless work at New Brunswick, N. J.

Several other methods have been devised for the production of high-frequency currents, such as Vreeland's "Sine Wave Oscillator," but they need not be considered in connection with electric heating, which requires large power units and at least moderately high electrical efficiency, neither of which requirements are capable of being met by the present development of these special methods.

In our furnace development, up to the time of this writing, we have employed the Tesla Oscillatory Current Circuit, fed with high voltage 60-cycle current, as our source of high-frequency current. For large power installations, however, the high frequency alternating current generator seems to give promise of being the most simple and practical source of high-frequency cur-

[5] Consult *Wireless Telegraphy,* by Dr. J. Zenneck, Chap. IX.

rent in large quantity. A test was made with a small Alexanderson alternator, giving 80,000 cycles per second. The heating which was obtained verified fully all the predictions of theory.

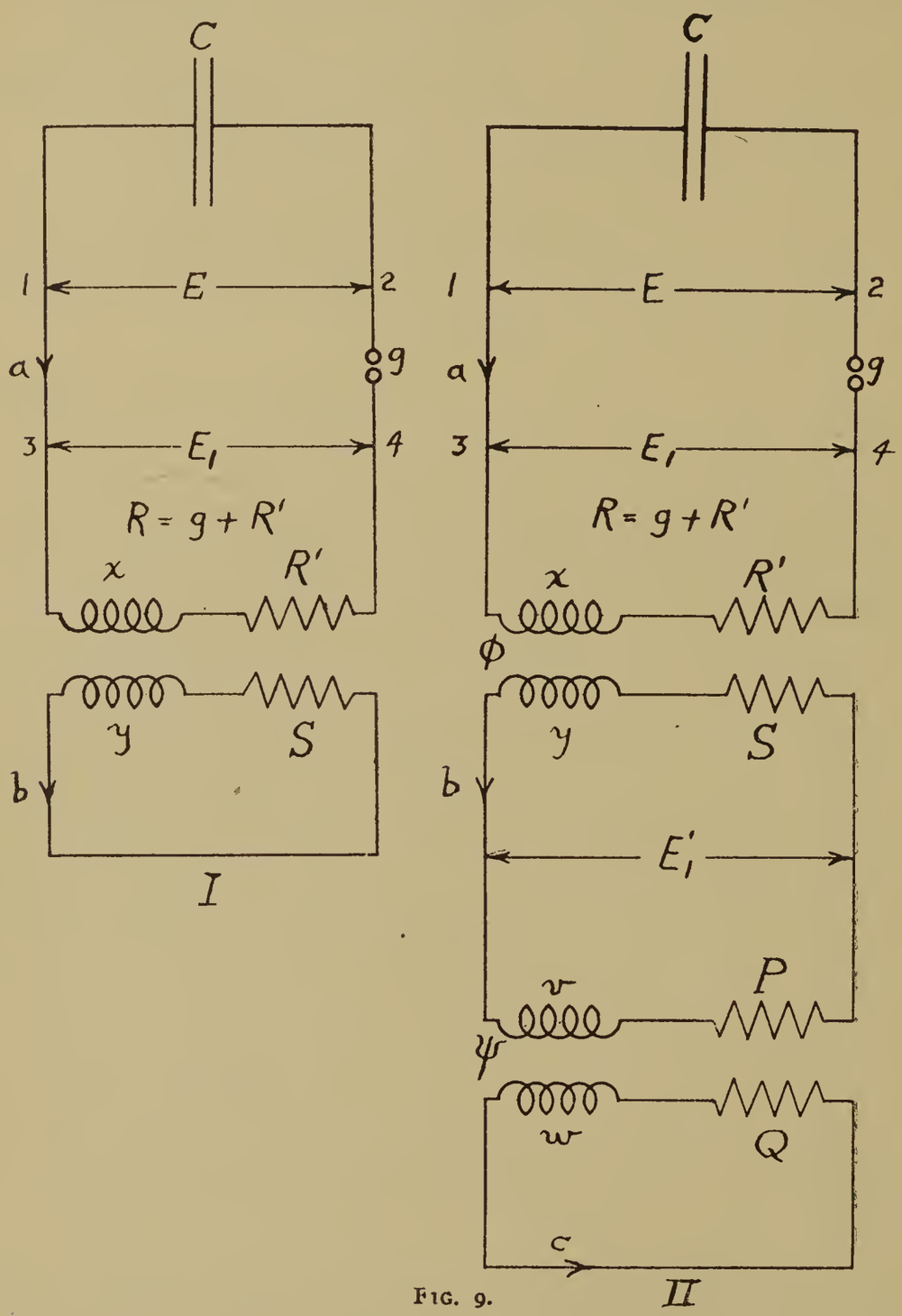


FIG. 9.

PRODUCTION OF HIGH FREQUENCY CURRENTS BY OSCILLATORY DISCHARGE.

In *I*, Fig. 9, is represented a single phase oscillatory current circuit magnetically coupled to a secondary circuit which has an ohmic resistance S in which it is desired to develop heat. In *II*,

Fig. 9, is represented an oscillatory current circuit which has a degree of coupling φ with a secondary circuit, and this in turn has a degree of coupling ψ with a quaternary circuit which has an ohmic resistance Q in which it is desired to develop heat.

We may consider that a sine-wave e.m.f. E is impressed in each case at points (*1*) and (*2*) on the condenser side of the discharge gap. If g is the average resistance of the gap (defining resistance as $\frac{\text{e.m.f.}}{\text{current}}$ then the average resistance of the primary circuit will be $R = g + R'$, where R' is the ohmic resistance of the primary coil. With this understood, all the equations which have been deduced in connection with Figs. 3 and 2 will apply to *I* and *II*, Fig. 9.

Or we may consider that the impressed e.m.f. E_1 between points (*3*) and (*4*) is given experimentally by means of an electrostatic voltmeter. In this case our equations deduced in connection with Figs. 3 and 2 also apply when we take R' to be the ohmic resistance of the primary coil.

It will be found by trial that E_1 is always much less than E; in one case, where a measurement was made, $E_1 = 1/6\ E$. When for any particular case E_1 is given experimentally, quite exact calculations of the magnitudes of the various currents are practicable. These currents, as also E_1, will be given as effective (square root of mean square) values and the rate of heating in S or Q will be the product of the square of this effective current by the resistance.

When energy is fed into the circuits by means of a given e.m.f. E_1, impressed at points (*3*) and (*4*), its final transformation into heat in either resistance S or resistance Q is best considered from the viewpoint taken in deducing the general equations in connection with Figs. 2 and 3. But when the e.m.f. E, impressed at points (*1*) and (*2*), is given, and the peculiar characteristics of the discharge gap causes a series of oscillatory discharges of the condenser C, then the progress of the energy transformations may be with advantage considered from other points of view. To these we now proceed.

I. Storage of Electric Energy.

The potential electric energy stored in unit volume of a dielectric of specific inductive capacity K and filling the space between two parallel conducting surfaces separated by unit distance and at a difference of potential V is,

$$E = \frac{KV^2}{8\pi} \tag{109}$$

For every dielectric there is a certain resistance to electric breakdown such that a potential gradient of maximum value V_m will just fail to produce a rupture. If we substitute V_m for V in Eq. (*109*) we have,

$$E_m = \frac{KV_m^2}{8\pi} \tag{110}$$

to express the maximum amount of electric energy which can be stored in the unit of volume of any particular dielectric.

E_m is very different for different dielectrics, and it is to be particularly noted that, while it varies directly with the specific inductive capacity it varies as the square of the near-to-breakdown potential gradient. Thus, when choice is made of a dielectric material with which to make a condenser to contain the maximum possible quantity of electric energy per unit volume, as much will be gained by the selection of a material of twice the rupture strength as by the selection of a material of four times the specific inductive capacity.

Further, when a large quantity of electric energy is to be stored in a dielectric it becomes important to select that dielectric material which will permit the storage of a given quantity of electric energy at a minimum money cost. If all parts of a condenser, other than its dielectric, are assumed to be negligible in cost, it is easy to find a guide to the selection of the most economical dielectric material. Thus,

$$\frac{1}{\frac{V_m^2 K}{8\pi}} = \frac{8\pi}{V_m^2 K} \tag{111}$$

are the number of units of volume of dielectric required to store a unit quantity of electric energy. Let P be the price in dollars

of one unit volume of a particular dielectric which is selected as a standard material. Then,

$$\$ = \frac{8\pi P}{V_m^2 K} \tag{112}$$

is the cost in dollars to store a unit quantity of electric energy in this standard dielectric material. Similarly for any other dielectric material of maximum rupture strength V'_m, of dielectric constant K' and of price P' dollars per unit volume, the cost of storing unit quantity of electric energy is,

$$\$' = \frac{8\pi P'}{V_m'^2 K'} \tag{113}$$

Thus we obtain (for the cost to store a given quantity of electric energy in any dielectric under consideration as compared with the cost of storing the same energy in some standard dielectric) the expression,

$$\$' = \frac{P'}{P} \cdot \frac{V_m^2 K}{V_m'^2 K'} \$ \tag{114}$$

If we apply this equation to determine the relative cost of using sheet glass to the cost of using mica in the construction of a condenser for holding a large amount of energy at high potential, it will be found, for example, that the cost of using mica is about ten times the cost of using glass.

II. Release of Electric Potential Energy.

Potential electric energy of necessity is always stored, not at a point in the dielectric, but is distributed throughout a volume of greater or less dimensions. The storage of electric energy means that the dielectric throughout a certain volume has assumed a state of electric strain.

Upon completing the connection between the conducting surfaces on which the electrostatic lines have their ends the strain in the dielectric immediately begins to diminish, at a great but not at an infinite rate. The originally stored potential energy, measured by $\frac{1}{2}CV^2$ rapidly transforms into kinetic energy, which, to a first approximation, is measured by $\frac{1}{2}Li^2$, where L is the coefficient of self induction in circuit and i the instantaneous

current. If the system is conservative—that is if it cannot lose energy by radiation, by dielectric loss or by heat—the potential energy becomes wholly transformed into kinetic energy $\frac{1}{2}LI^2$, where I is the maximum instantaneous current, and then immediately transforms back again into potential energy, and thus there occurs an oscillation of energy between the one form and the other. This oscillation of energy between the forms $\frac{1}{2}CV^2$ and $\frac{1}{2}LI^2$ would continue indefinitely were it not for unavoidable losses. If the conducting, oppositely electrified, surfaces of the capacity are of considerable area, there will exist a measurable self induction taken between any two not-near-points on either of the conducting surfaces.

The conductor used to join the two conducting surfaces of opposite polarity likewise possesses self induction. Thus, if we conceive either of the conducting surfaces divided into a number of imaginary small equal areas it will be seen that the dielectric strain under each of these small areas will be relieved (when oppositely charged conducting surfaces are joined), through conducting paths which differ in their self induction. In other words, each imaginary small condenser, taken in connection with the particular self induction in series with it, forms an oscillatory system which has its own particular period, and this is never exactly the same for any two of the systems. As a consequence of this the surge of energy between the potential and the kinetic form can never be purely sinusoidal and there will always be superimposed, to a greater or less degree, upon the fundamental oscillation of the entire system certain higher harmonics. Furthermore, local surges of greater or less amplitude will occur between different parts of the system, which, in rapidly damping out, mean local dissipation of energy in dielectric losses, in I^2R heat losses and in radiation losses.

It would appear from considerations of this character that it is impossible to produce strictly pure sinusoidal oscillations of electricity or of electric energy, which for their exact realization would require a capacity localized at a point.

III. Practical Methods for Reducing Local Surges.

It is not uncommon to speak of an electrostatic capacity as either "bulked" or "distributed," but strictly speaking all electro-

static capacity, to a greater or less degree, is distributed capacity. The different elements of a condenser cannot be physically connected so any two small equal elements will receive an electric charge or release one in exactly equal time-intervals. Thus at any given instant certain elements of the condenser will hold a different charge and have a different potential than other elements, and local oscillations are established which dissipate energy within the condenser, and within conductors which join its different elements.

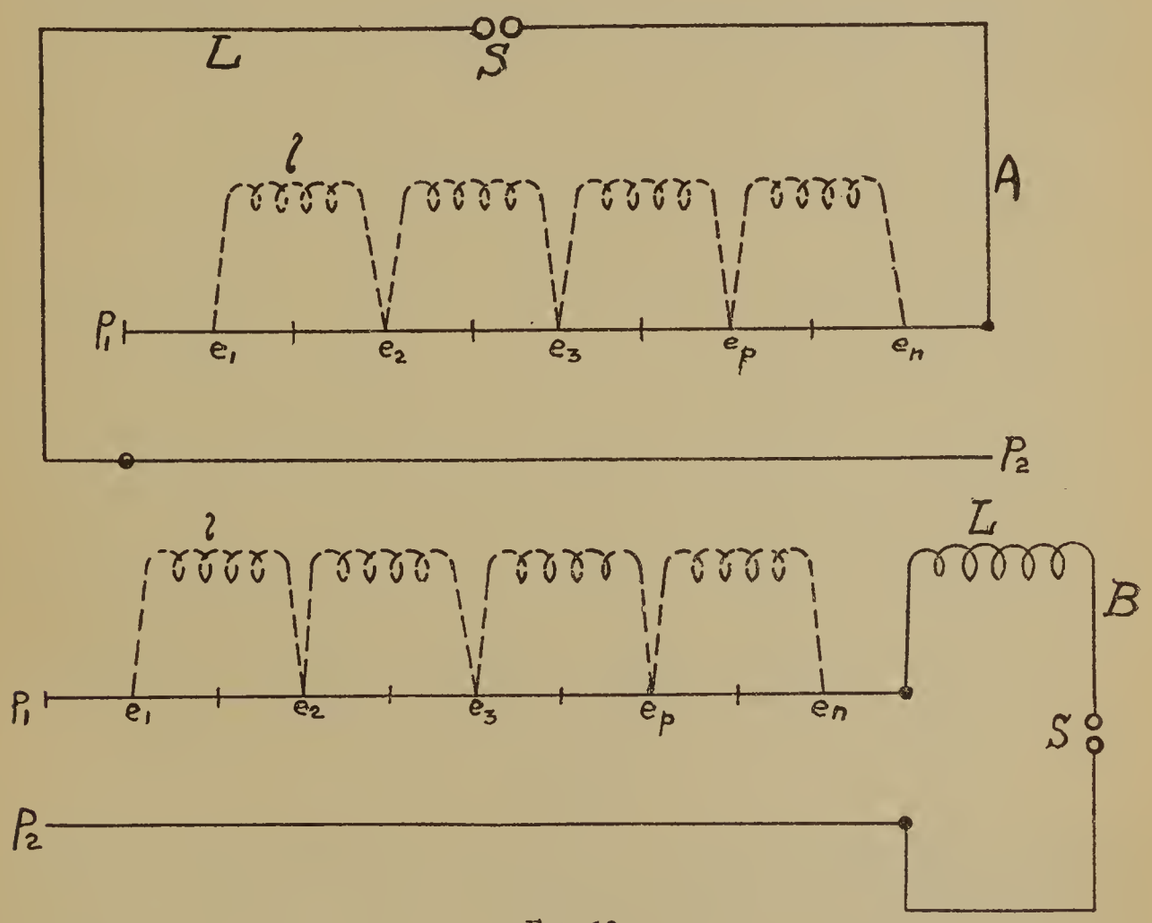


FIG. 10.

To utilize to a maximum extent the potential electric energy which may be stored in an electric condenser; for the production of electric radiation, chemical action, or light, or localized heating, it is necessary to so construct individual condenser units and to join these units in such manner that the system, consisting of capacity and inductance, will oscillate, nearly as may be possible, as a single system. A large increase in efficiency may be obtained by arranging to avoid as much as may be the local surgings which produce energy dissipation and result in unutilizable energy transformations.

This matter is of such large importance in the practical utilization of stored electrical potential energy that we shall consider it further by examining some special cases which are typical.

In Fig. 10, p_1, p_2 of A and B may be considered conducting strips which constitute the coatings of a long narrow plate-condenser. The condenser thus formed may be discharged through a discharge gap S with conductor-leads L joined to the plates p_1, p_2, as in A or as in B. We may conceive the conducting strips, p_1, p_2, as made up of several elements as e_1, e_2, e_3, etc., and we may further conceive that from the center of one element to the center of an adjacent element there exists a certain self induction

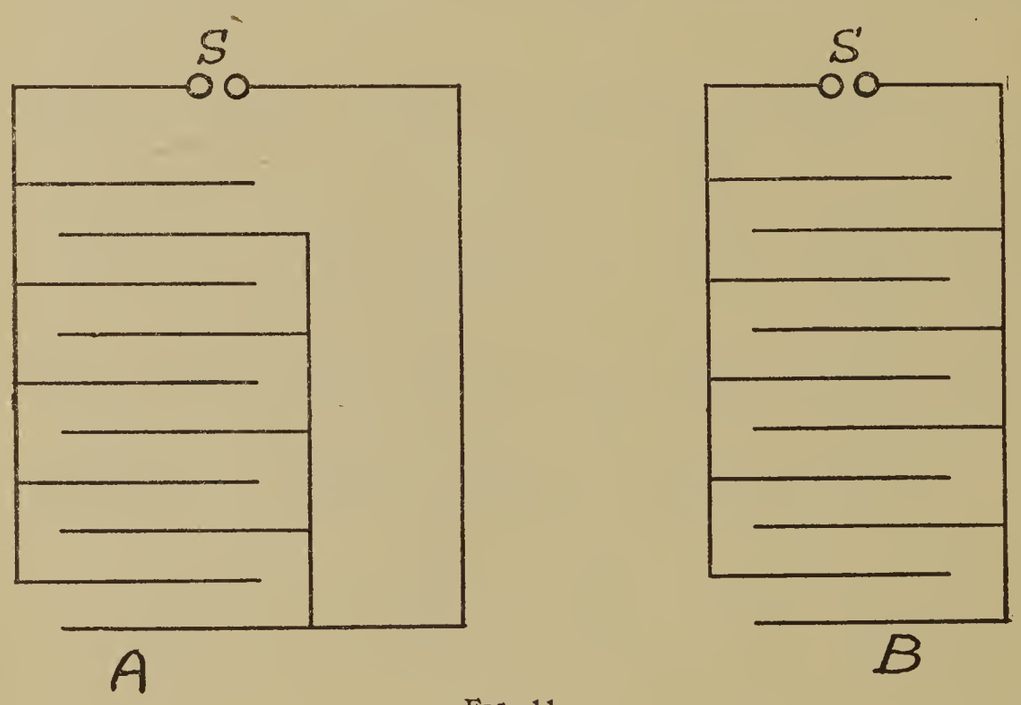


Fig. 11.

l (indicated by dotted lines). It at once appears that the connection shown at A is better than the connection shown at B, for avoiding local surging. Thus, in the connection B, upon closure of the gap S, the condenser elements e_1, e_2 . . . e_n, begin to discharge, but the rate of discharge of e_n exceeds that of e_1 because e_1 must discharge through a greater inductance than e_n. Hence, after a brief interval the element e_n is at a lower potential than the element e_1, and this, therefore, will discharge into e_n, and local surging of the electricity will thereupon be set up between these two elements and to a less extent between other pairs of elements not so far separated.

In the connection A on the other hand, any element as e_p, for

example, in discharging must discharge through the same total inductance as any other element, and, consequently, there is less tendency to local surging. However, as all parts of either the strip p_1 or p_2 do not carry the same current, the various elements e_1, e_2, etc., do not change in potential at precisely the same rate and the tendency to local oscillations is not wholly eliminated, though this tendency is much less in connection *A* than in connection *B*.

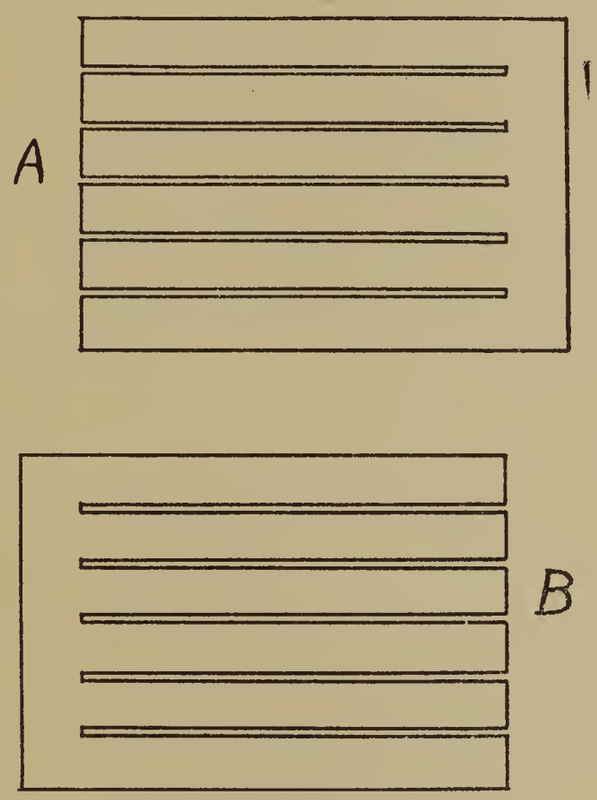


FIG. 12.

The reactance between any two oppositely charged elements of the condenser becomes greater as the natural oscillation frequency of the system, taken as a whole, is greater and this frequency is increased when the inductance *L*, external to the condenser, is diminished. This consideration leads to the conclusion that energy losses which result from local surging will diminish when the inductance external to the condenser is increased.

In making connection to a stack of condenser plates, diagonally opposite edges should be joined to the gap *S* as in *A*, Fig. 11, and not directly opposite edges as in *B*, Fig. 11.

Furthermore, if the inductance external to the condenser is small and the capacity not large, so the system as a whole oscil-

lates with a high frequency, it is desirable to cut slots in individual condenser sheets in the manner shown in Fig. 12, *A* and *B*. Plate *A* is superimposed over plate *B* when the condenser plates are laid up.

By so doing and by connecting plates of opposite polarity at symmetrically located points surges from side to side of individual sheets may be almost wholly avoided.

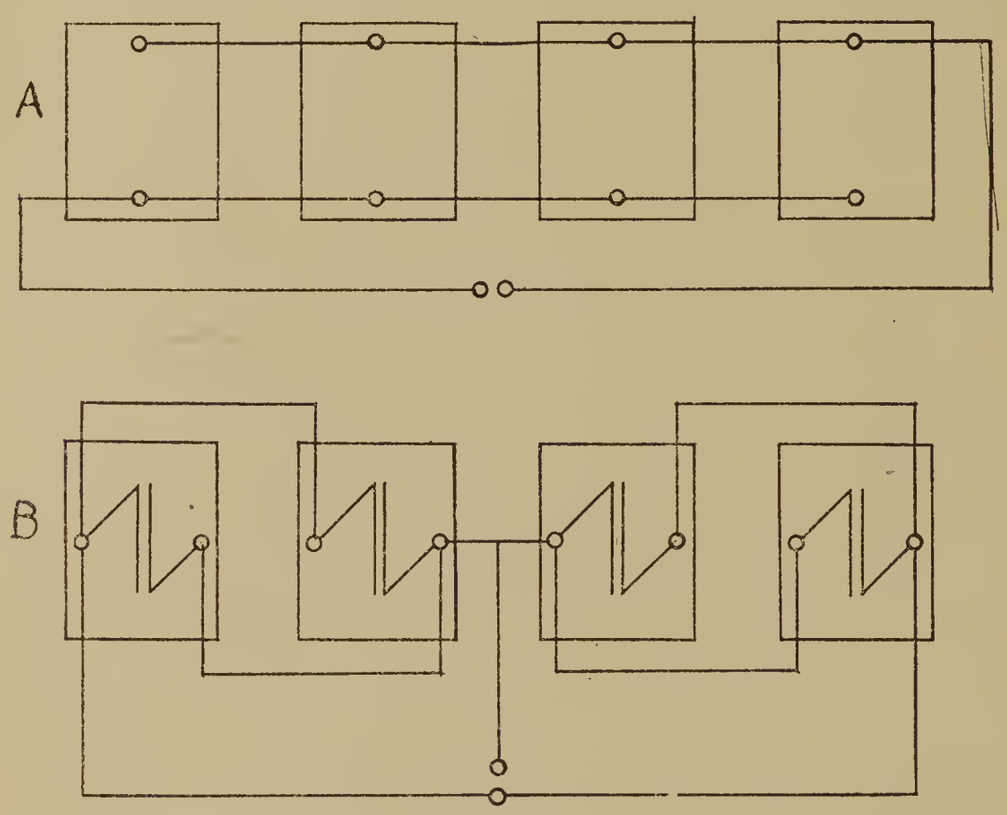


Fig. 13.

When a condenser is constructed for the storage of a large quantity of energy a large mass of dielectric material must be used. Practical considerations require the subdivision of the mass of material into several or many units. In other words, several or many condenser units must be joined in parallel to give the required energy-storing mass. For avoiding local oscillations between individual units each unit should have a like construction, size and electric capacity. In joining several units to form a group, constituting a single condenser of large capacity, the most careful consideration should be given to the manner in which the individual units are joined. The general principle to follow is to arrange the connections so that, as nearly as possible, each unit in the group must discharge through the same total

inductance. The principle stated in connection with Fig. 10 and Fig. 11 should be observed also when connecting together several units to form a group.

In Fig. 13 are shown two proper methods of joining four plate-condenser units to form a single condenser. The same principle,

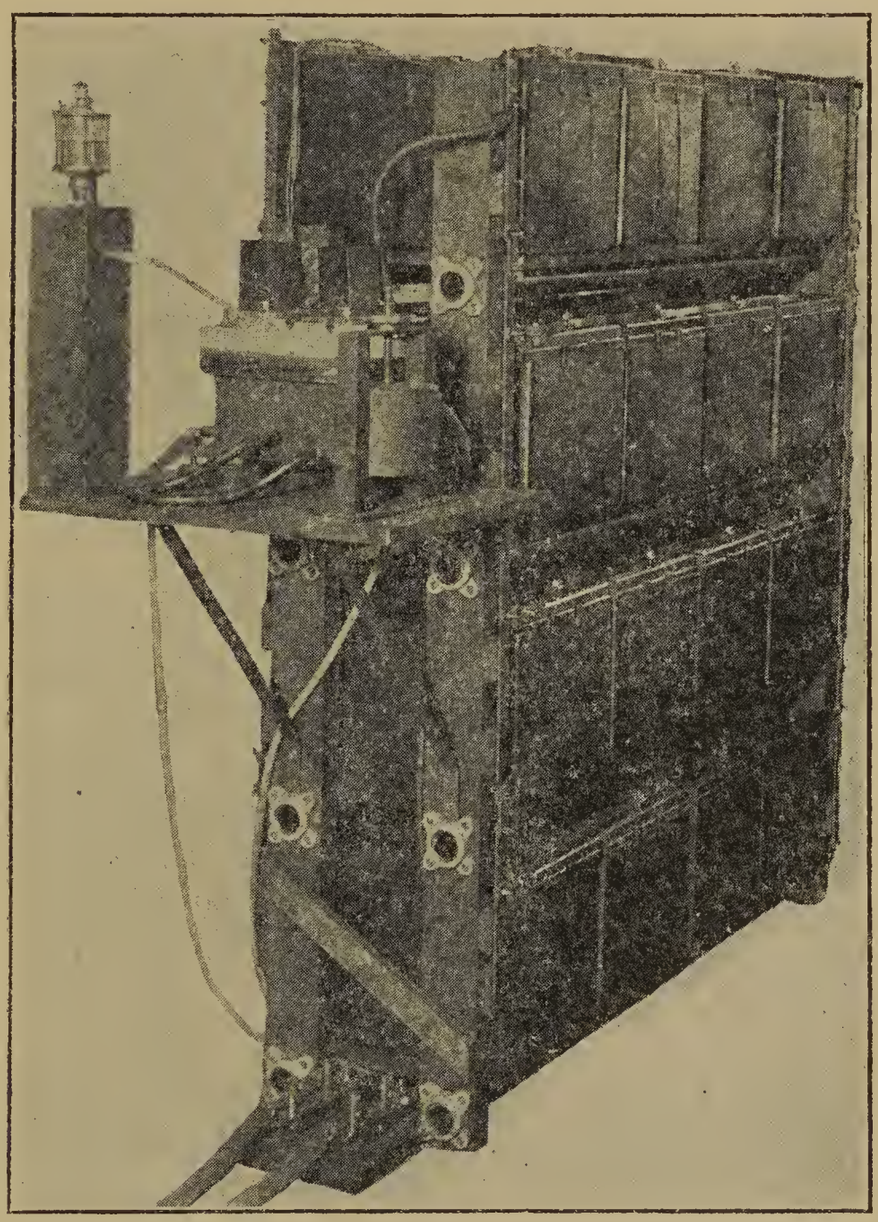


Fig. 14.

illustrated in Fig. 13, is capable of extension to any number of condenser units.

In Fig. 13 the grouping *A* is the simpler method to employ, but it is not thought to be quite so good theoretically as the grouping *B*.

In Fig. 14 is shown, in the photographic reproduction, the

grouping adopted by the writer for a bank of sixteen condenser units which is capable of absorbing 22 kilowatts from a 60-cycle, 6,000-volt supply.

IV. Conditions Suitable for the Production of Successive Oscillatory Discharges; An Effective Discharge Gap.

When a charged condenser has been discharged and its stored energy has become transformed into useful heat, it is necessary, for the continuous operations, to repeat the charging and discharging at frequent regular intervals. If N' is the number of times stored potential energy is transformed in the unit of time, the power absorbed and transformed is simply the product of the average value of the stored potential energy by the number of times N' this energy is transformed in unit time. For power utilization at 100 percent efficiency we have therefore the formula,

$$P = EN' = \tfrac{1}{2}CV^2N' \qquad (115)$$

A large portion of the power supplied to a condenser may be measured by changing it into localized heat. It is never possible, however, to take account of all the kinetic energy into which the stored potential energy is transformed when discharges of the condenser take place. Hence, if we call k the fractional part of the stored energy which can be transformed into localized, measurable kinetic or heat energy, the expression for power which may be utilized is,

$$P_m = \tfrac{1}{2}kCV^2N' \qquad (116)$$

Into whatever form the energy, which may be stored in a condenser, is to be converted, it is important to so choose conditions that both k and N' are large as possible. The success obtainable in this respect depends largely upon the methods used for successively charging and discharging the condenser.

The customary types of discharge gap, supplied with an air blast for reducing arcing at the gap, are inadequate for various reasons, and a special new type discharge gap has been devised which performs the required functions of an oscillatory current discharge gap in a highly perfect manner and permits the production of powerful electrical oscillations of great regularity.

This new gap is designed to operate with high voltage alternating current supply.

The manner in which the ordinary forms of discharge gap operate may be explained as follows: Referring to Fig. 15, it will be observed that when the gap *g* is completely open the con-

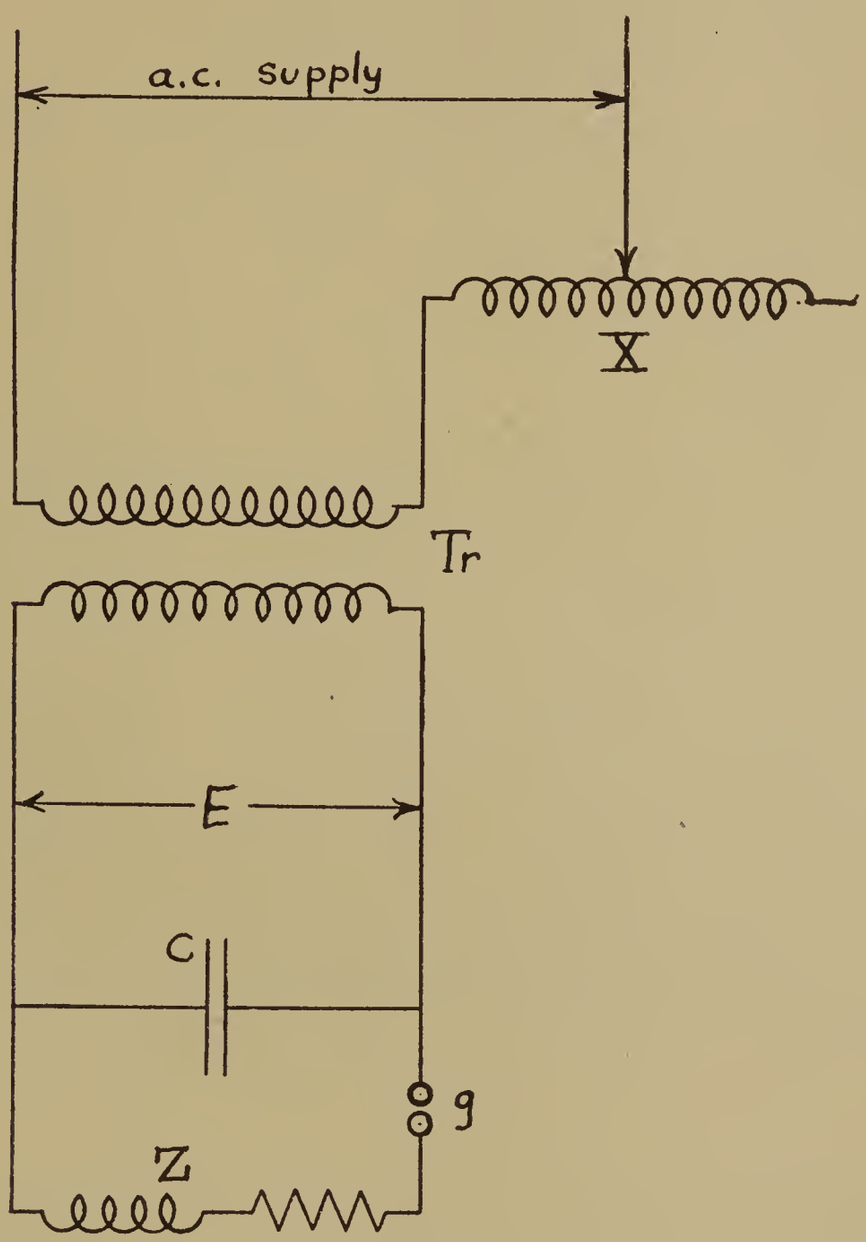


FIG. 15.

denser *C* will continue to charge, while the secondary alternating voltage *E* continues to rise. When *E* attains a magnitude which ruptures the gap *g* the condenser *C* discharges through the gap and the impedance *Z*; the discharge being oscillatory if the energy is not too rapidly dissipated. The moment, however, that discharge through the gap *g* starts, the resistance of the gap suddenly falls

to a low value and the secondary of the supply transformer *Tr.* is short-circuited through the gap. Were it not for the reactance of the transformer itself and additional reactance *X*, generally placed in series with either the primary or secondary of the transformer, a very large current, supplied by the transformer, would flow through the gap. But when a large amount of power is being handled, even with reactance to limit this current, an arc tends to form across the gap which highly ionizes and renders conducting the dielectric, gas, liquid or solid, between the gap

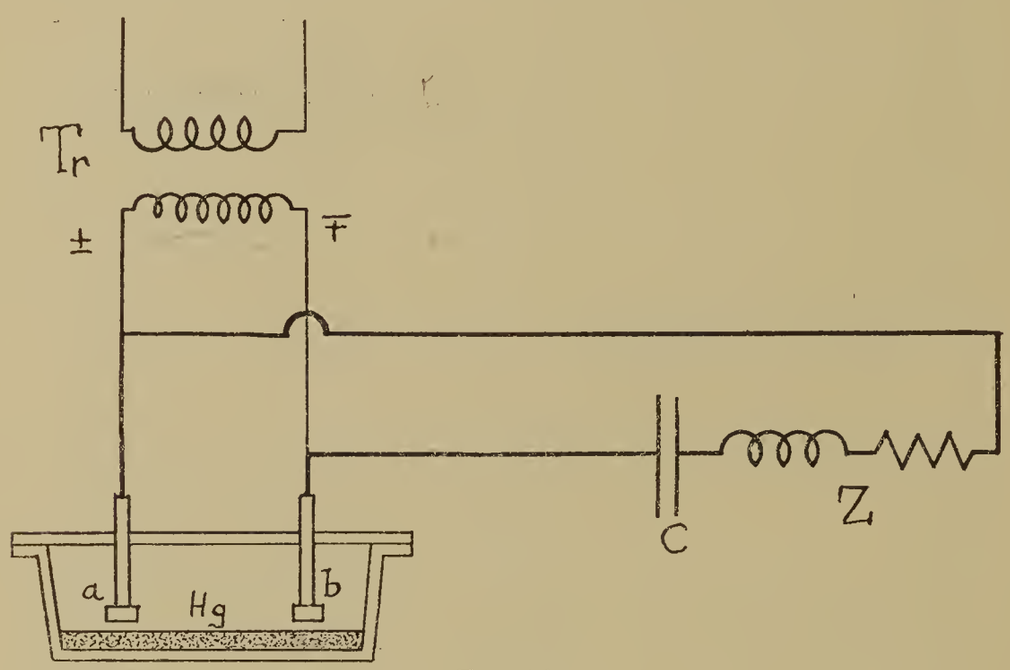


Fig. 16.

terminals. Consequently the voltage drop over the gap does not again rise, the condenser is but little charged the second time and no continuous series of oscillations is established. To prevent an arc it is customary to use a powerful air-blast to drive the ionized air or vapor from between the gap terminals and thus to bring up the gap-resistance and so permit the continued recharging of the condenser. It is also usual to assist the air-blast in dispelling the ionized dielectric by rotating the gap electrodes at high velocity in reverse directions.

All the results gotten with rotating electrodes and an air-blast may be improved and more simply obtained with the new gap, to be described, which utilizes the extraordinary property possessed by pure mercury, known as its "negative electrode resistance," a property which it possesses not alone when in a vacuum

but at atmospheric pressure, and, to a greater or less extent, in various gases and vapors. The arrangement employed for utilizing this property of mercury is shown diagrammatically in Fig. 16.

It is to be noted that, in which ever direction the current from the secondary of the transformer is following, this current, in passing through the double gap, must make one passage from

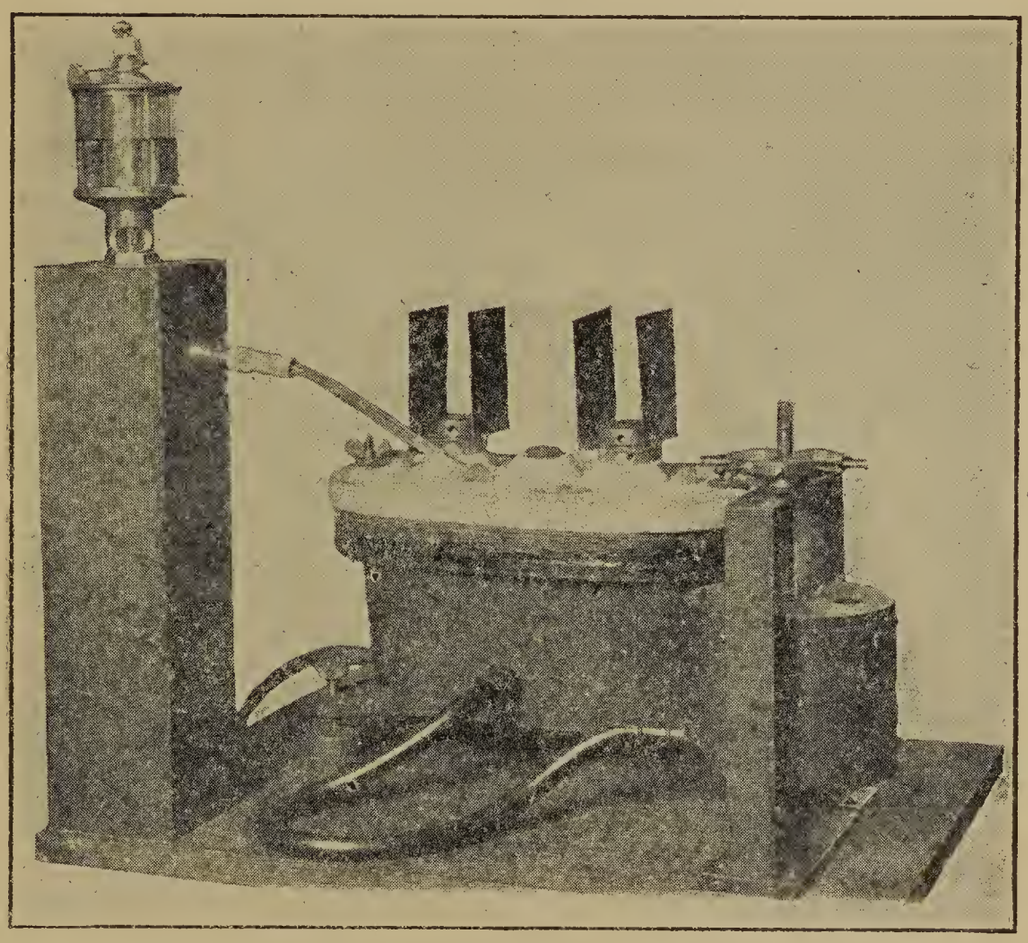


Fig. 17.

mercury to either one or the other of the solid electrodes. Experiment shows that regardless of the nature of the solid electrodes when the voltage between mercury and one solid electrode attains a certain minimum, the negative electrode-resistance of the mercury acts to completely open the circuit, and with great suddenness. No arc forms, nor can one be maintained. Consequently, after a discharge has ceased, the voltage across the condenser terminals rapidly rises again to its full value, the condenser again becomes charged, the double gap again ruptures, oscillations occur

and the process is thus repeated in very rapid and regular succession. While the discharge is taking place a very localized high temperature is developed in the gap. This maintains its resistance very low and the discharge, oscillatory or otherwise, occurs with relatively little energy dissipation in the gap itself, provided energy is being elsewhere absorbed in the circuit. Nevertheless, when the oscillations have nearly died out, the gap automatically and very suddenly completely opens, thereby shutting off any flow of current from the supply transformer, and all arcing is thus absolutely prevented.

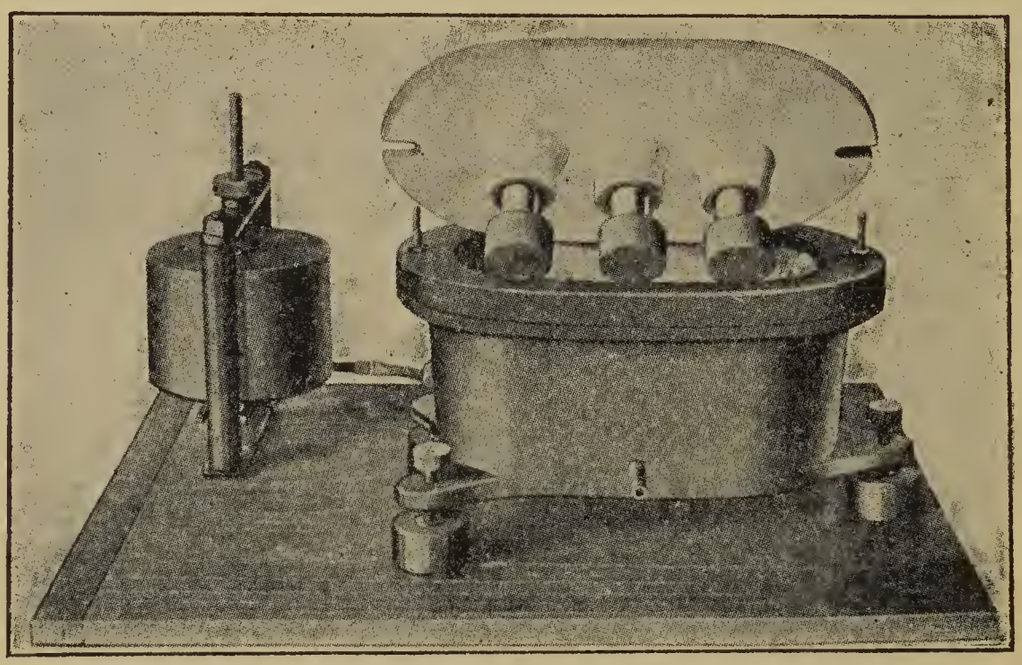


Fig. 18.

This remarkable property of mercury is enhanced and the production of chemical action of any kind on the mercury is wholly prevented, if the double gap be placed in a container filled with the vapor of alcohol.

A gap of this character, designed to handle from 20 to 30 K.W., is shown in the photographic reproductions, Figs. 17 and 18.

Fig. 17 is an assembled view of the gap. In Fig. 18 the cover is shown raised to expose the construction of the electrodes. The expanded electrode tips are of Acheson graphite and are made removable by unscrewing. The flat pieces of copper attached to the electrodes are heat radiators. The level of the mercury in the gap is changed and the gap lengths adjusted by raising or lower-

ing the mercury reservoir shown on the right in Fig. 17. Alcohol is admitted at the rate of about 20 drops per minute from the dropper shown at the left, Fig. 17.

This form of gap is practically noiseless. There is no consumption of the mercury, the surface of which is maintained clean and bright, provided an alcohol atmosphere is maintained in the gap. The gap has no moving parts and therefore requires no attention and has nothing to wear out and the oscillations obtained with it are strong, maintained and of great regularity. The gap can be used for either single-phase or three-wire, two-phase operation, the one illustrated being provided with three electrodes for two-phase operation.

V. Power Condensers.

Many practical considerations, such as ease of construction, compactness of form and minimum cost of dielectric to store unit quantity of electric energy led the writer to adopt sheet-glass for the construction of high-potential power condensers where continuous operation not exceeding 3 to 5 hours is contemplated. The glass and sheet-metal plates are stacked and kept under oil to prevent energy losses from brush discharge.

Only that portion of the sheet-dielectric is effective for storage of energy which lies between two opposing conducting surfaces of opposite polarity. By Eq. (*116*) the power which may be obtained from a power condenser, in a manner to be utilized, is $P_m = \frac{1}{2}kCV^2N'$. If the condenser is made of n sheets of dielectric and $n + 1$ sheets of metal its capacity is, $C = \frac{sKn}{4\pi d}$ when expressed in electrostatic units. If expressed in Farads, the capacity is,

$$C_f = \frac{sKn10^{-13}}{1.131d} \qquad (117)$$

where s is the effective area in square centimeters of one sheet of dielectric, d the thickness in centimeters of any one sheet, K the specific inductive capacity of the dielectric, and n the number of sheets of dielectric used.

The maximum safe charging voltage V_m which may be applied to this condenser will depend upon the strength per unit thickness

of its dielectric to resist rupture and upon its thickness. We may therefore write $V_m = Ad$, where A is a constant determined by the nature (and incidentally by the temperature) of the dielectric. Thus, we may write, $V^2 = V_m^2 = A^2d^2$. Putting this value of V_m^2 and the value of C_f in the above equation for the power we have,

$$P_m = \frac{kKN'A^2}{2.262} \text{ snd } 10^{-13} \qquad (118)$$

But *snd* is the total volume v of the dielectric which is effective for storing energy, and so we have,

$$P_m = \frac{kKN'A^2}{2.262} v10^{-13} \text{ watts} \qquad (119)$$

This is an expression for the maximum utilizable power obtainable from any power condenser. The equation shows that the power depends directly on the effective volume of dielectric used, on factors K,A^2 which are determined by the nature of the dielectric material used and on the factors k,N' which are, in large measure, determined by the design of the condenser and the oscillation circuit in which it is used.

By placing $d = 1$ cm., the relation $V_m = Ad$ defines A as the maximum voltage which may be applied safely and continuously without producing rupture to 1 cm. thickness of the dielectric used. No condenser has more security against breakdown than is given by the strength against rupture of its weakest plate. The value of A will be considerably less, therefore, than the value which might be obtained from tests made on single plates. Moreover, a rise in temperature (brought about by continuous service or otherwise) of from 20° to 30° C. will much reduce the value of A. Furthermore, a generous factor of safety should be allowed when continuous and uninterrupted service is contemplated.

When consideration is given to the above, experience shows that the value of A, for a good quality of window glass selected to be free from noticeable defects and uniform in thickness, may be taken as 50,000 volts per cm. By good designing the value of k may be chosen as high as 0.8, and with oscillating currents

the value of K is 5.5. With 60-cycle supply, even a large condenser may be charged twice in each half cycle. Thus $N' = 240$. Using these values and placing $v = 1$ cm.^{3}, Eq. (*119*) gives,

$$P_m = \frac{0.8 \times 5.5 \times 240 \times \overline{50{,}000}^2 \times 10^{-13}}{2.26} = 0.1167 \text{ watt}$$

This is the power which, assuming no rise in temperature, may be obtained continuously from 1 cm.^{3} of window glass. It will be a very conservative figure to take 0.11 watt per cubic centimeter of ordinary window glass.

We may now inquire what power may be obtained, using 60-cycle supply, from a condenser unit of convenient size which fills the following specifications in regard to its dielectric.

Dielectric, best quality window glass.
Length of sheet, 35.5 cm. (14″).
Width of sheet, 27.9 cm. (11″).
Thickness of sheet, 0.317 cm. (0.125″).
Number of sheets, 60.
Total volume of glass, 18,840 cm.^{3} (1,152 cu. in.).

By a correct choice of dimensions of the 61 metal sheets used, the total effective volume of this glass will be 13,140 cm.^{3} (804 cu. in.). The utilizable power obtainable is,

$$13{,}140 \times 0.1167 \times 10^{-3} = 1.538 \text{ kilowatts.}$$

This result has been fully established by experiments.

There exists, however, a serious objection to the use of glass as a dielectric for condensers which are intended for long, continuous operation. This objection rests on the unfortunate property possessed by glass, when subjected to a high-frequency variation of potential, of not only heating but of heating at an increasing rate as the temperature rises. Experience shows that a condenser, meeting the above specifications, will operate continuously, without undue heating, from three to four hours, under oscillatory current conditions. By the end of this time the entire mass of plates will have risen to, perhaps, 50° or 70° C., and the plates will begin to rupture under an electric stress which they will stand well when at 20° C. However, the weakening

of the plates is not permanent, for when the condenser is allowed to cool again it regains completely its electrical strength. There is no evidence of any progressive weakening, from intermittent use of the condensers, over long periods of time. But since it takes nearly as long for a box of condenser plates to cool as it requires to heat, it is necessary, if the operation of the apparatus must be continuous, to provide two sets of condensers, so one set may be cooling while the other is in use. Artificial cooling has not been tried but it would undoubtedly prove effective if the complication and expense justified it.

The General Electric Company have standardized and manufacture in large quantities power condensers which make use of paper for the dielectric and aluminum sheet 1 mil (0.025 mm.) thick for the metal plates. The built-up structure is strongly compressed beneath oil with vacuum above. This built-up structure is placed under oil in a metal case and hermetically sealed. These condensers sell for about twice the cost of glass, condensers capable of absorbing an equal power, but tests made by the writer show that the heating, under oscillatory current conditions, is so slight that the condensers reach an equilibrium temperature which is well below the temperature which weakens the dielectric. A temperature rise of not over 10° C. may be expected, and it is to be hoped that these condensers will prove entirely satisfactory and reliable when placed in continuous service under oscillatory or high-frequency current conditions; at least this has proved to be the case in connection with a sample unit which was carefully tested by the writer for performance when kept in continuous service at full power load for twelve hours. The condenser units as supplied by the General Electric Company have, furthermore, the advantage that they may be shipped in completed form, while condensers built up of glass and metal sheets under oil must be assembled where they are to be used. The glass condensers, on the other hand, have the great advantage that a ruptured plate may be readily replaced, while a rupture in a paper condenser practically destroys it.*

* Since the presentation of this paper the writer has been able to obtain mica condensers from the Dubillier Condenser Co., at reasonable prices. Condensers of this type greatly reduce the weight and bulk of the installation of condenser-units. They permit of continuous operation with a temperature rise in the condensers of less than 3° C. They are perfectly adapted for installations in small power units, and several such installations are now in apparently satisfactory service in customers' hands.

VI. Forms Into Which Electric Potential Energy May Be Transformed.

The energy stored in a condenser when released may be transformed partly or largely into a variety of forms as determined by the conditions imposed. This transformation may occur in a single direct process or in several stages. Thus, the energy may be converted into: sound or noise, light, electron acceleration and X-rays, visible mechanical motion, invisible molecular motion or heat, chemical action either in combining or in dissociating chemical elements, or electric radiation.

Since the law of conservation of energy holds here as elsewhere the energy transformed into any particular form plus the energy transformed into all other forms must exactly equal the original supply of stored energy. The average rate or velocity of the conversion into energy of a particular type is the average power as manifested in this type. If the energy transformations constitute a uniform continuous series, as when an oscillatory system is fed with constant power, then, for transformations into any particular type of power, the efficiency of the entire system is the utilizable portion of the particular type of power developed divided by the power supplied to the condenser at its terminals. Thus the efficiency of an oscillatory system, designed to emit radiation from an antenna, would be the ratio of the actual power radiated from the antenna (exclusive of the radiation from the leads, etc.) to the power supplied at the terminals of the energy-storing capacity used.

Though it is possible to obtain considerable efficiency of transformation into several of the types of energy noted above, the writer is not aware that much has been done heretofore, using oscillatory currents, toward obtaining an efficient conversion of energy into any other form than that of electric radiation as employed in radio work. The writer has studied the transformation of stored potential electric energy, using oscillatory currents, into light (high-frequency arc light) chemical action, and more particularly into localized heat.

For the purpose of obtaining high temperatures, or rapid heat production in a mass of material, as obtained in a crucible or otherwise, oscillatory currents are particularly suitable for small and medium power installations. All the results sought may be

obtained without the use of iron or other magnetizable materials, and no magnetic circuit need interlink with an electric circuit, even when high efficiency in energy transformation is sought.

In what follows we shall confine our study to processes for the efficient transformation of stored electric potential energy into localized heat, this to be used for obtaining high temperatures and the heat treatment of material.

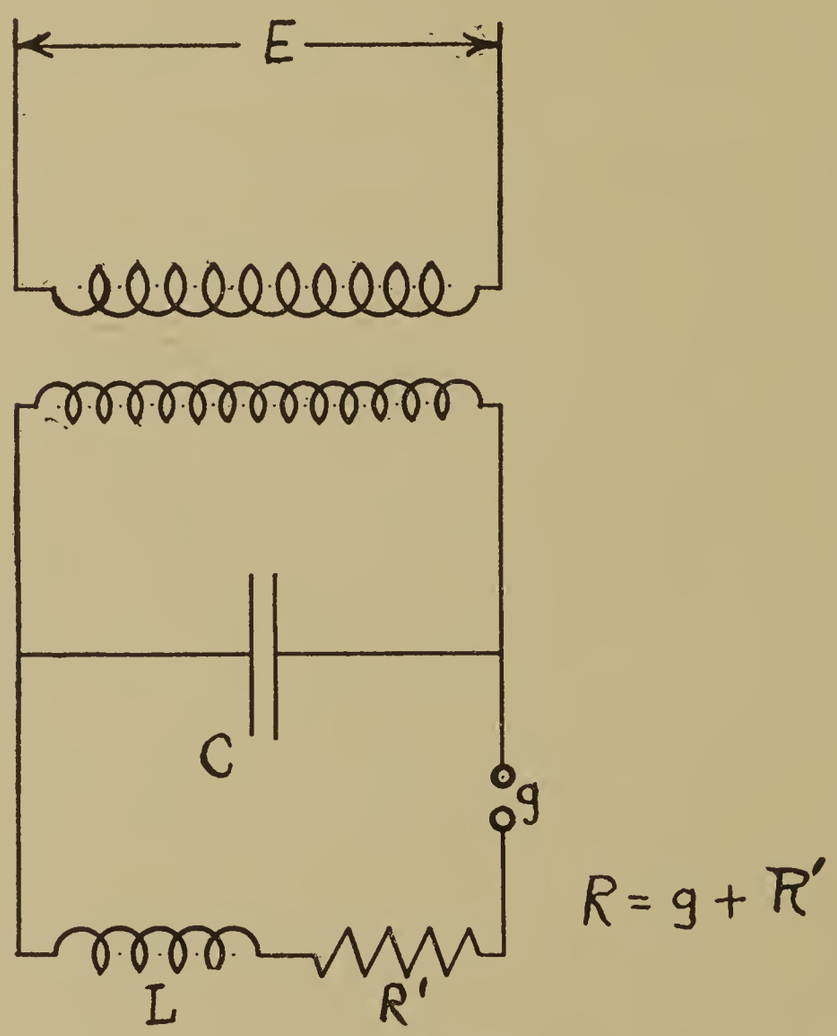


Fig. 19.

VII. Power Transformation Into Heat in a Single Oscillatory Current Circuit.

The simplest form of an oscillatorv current circuit is that shown in Fig. 19.

We may assume the conductors are so arranged or the frequency so low, or through both conditions being combined, that dissipation of energy by radiation is wholly negligible. With this assumed all the energy stored in the condenser C will, upon rup-

ture of the gap g, become transformed into heat. Represent by g the ohmic resistance of the gap g and all parts of the circuit wherein heat may be developed which cannot be utilized, and by R' the ohmic resistance of that portion of the circuit in which it is desired to develop heat.

We wish to inquire: what are the conditions which determine the rate of heat production in R' and what conditions determine the efficiency of the system as a converter of power supplied to C into power expended as heat in R'?

If the condenser C receives a single store of energy and then, on rupture of the gap g, releases this energy this will evidently be transformed into heat in the least possible time when the sum of g and R' is such that the discharge is just non-oscillatory, for this is the time of quickest complete discharge. This will be when

$$R' + g = R = 2\sqrt{\frac{L}{C}}$$

(See Bedell and Crehore, *Alternating Currents,* p. 108 *et seq.*) Now g may be made small but can never be zero. Hence to transform a portion of the stored energy in C into heat in R' in the shortest time we should make $R' = 2\sqrt{\frac{L}{C}} - g$

It is evident also that to convert the largest portion of energy stored in C into heat in R' we should make the ratio $\frac{R'}{g + R'}$ as large as possible. Hence efficiency of energy transformation into heat where required and rapidity of transformation are not to be obtained in the same way.

In power transformations, as made with oscillatory currents, as soon as C has released its stored energy it again acquires from the supply source a new store of energy. When this process is repeated N' times per second the absorbed power is evidently $P = \frac{1}{2} CV^2N'$. It follows that if we wish a maximum conversion of power into heat N' must be made as large as possible. If we make the assumption that the condenser, on account of the characteristics of the discharge gap, begins to receive new energy at the moment a certain portion of that previously supplied has been changed into heat, then for N' to be large the con-

version into heat of each single store of energy should be made rapid as possible, and this again will be when

$$g + R' = R = 2\sqrt{\frac{L}{C}}$$

If we call T the time required, after the gap ruptures, for a practically complete conversion into heat of the energy stored in a single charging of the condenser, and t the time required for the potential of the condenser to be again raised by the source of power from 0 to V, N' will be equal to $\frac{1}{T + t}$

The efficiency of conversion of power obtained from the source into utilizable heat developed in R' will be

$$F = \frac{i^2R'}{i^2g + i^2R'} = \frac{R'}{g + R'}$$

But if the condition has been filled whereby the maximum amount of power supplied is transformed we must have

$$g + R' = 2\sqrt{\frac{L}{C}} \quad \text{or} \quad R' = 2\sqrt{\frac{L}{C}} - g$$

in which case

$$F = \frac{2\sqrt{\frac{L}{C}} - g}{2\sqrt{\frac{L}{C}}}$$

Thus the efficiency can only become 100 percent when g is 0. Rapidity of conversion of power into heat in R' and efficiency of conversion are not obtainable by like conditions.

One is not, however, able to say, without experimental trial, to what minimum voltage the condenser must discharge before charging can begin again. It may well be that because of the peculiar characteristics of the mercury discharge gap recharging of the condenser begins before all of its charge, previously acquired, is lost. Indeed certain observations seem to indicate that when the condenser has so far discharged that its voltage has fallen to, perhaps, 2,000 volts, the resistance of the gap suddenly becomes infinite and recharging of the condenser at once begins

again. If this is so, N' may be made considerably larger than calculation, based on the assumption that the discharge is nearly complete before recharging begins, would indicate. If such be the case the absorbed power and the power convertible may exceed that based on the usual estimates. A fact of experience which may bear on the matter is that with a given gap length there is a certain critical supply voltage which causes the system to absorb more power than a higher voltage.

It should be recognized, in this method of power conversion into heat by oscillatory discharge, that we change power in the supply circuit, expressed by the product of a small current, quite or nearly in phase with its high voltage, into power in the oscillatory circuit expressed by the product of a large current, a high voltage and the cosine of a large angle of phase displacement between the two. Thus, a small wire, as a No. 32 copper wire, in the supply circuit will guide power into an oscillatory circuit in which a current of 50 amperes or more may flow and produce very striking heating effects. Thus if the resistance R' Fig. 19, consists of a gap between metallic electrodes, graphite, or conducting terminals of any kind, one readily produces a very brilliant, very steady and self-starting arc. The mercury discharge gap maintains steady oscillations in the system, at the same time preventing the supply circuit shorting through the arc. Applications to spectroscopic and other work are here suggested.

VIII. Circuits Used for Heating with High Tension Single-Phase Oscillatory Discharge.

In Fig. 20 are shown the circuits, which extensive experimentation and theoretical considerations have demonstrated are best adapted to give good results by inductive heating with oscillatory currents, when the source of supply is a single-phase circuit.

It should be noted that a reactance x is shunted across the primary terminals of the transformer and another reactance x_s, which is adjustable, is in series with the primary of the transformer. The function of the reactance x_a is to adjust the phase of the current I in the secondary of the transformer and thereby improve the oscillations and power consumption, especially where large amounts of power are demanded. The theory of the use of this reactance and how it may be expected to improve the

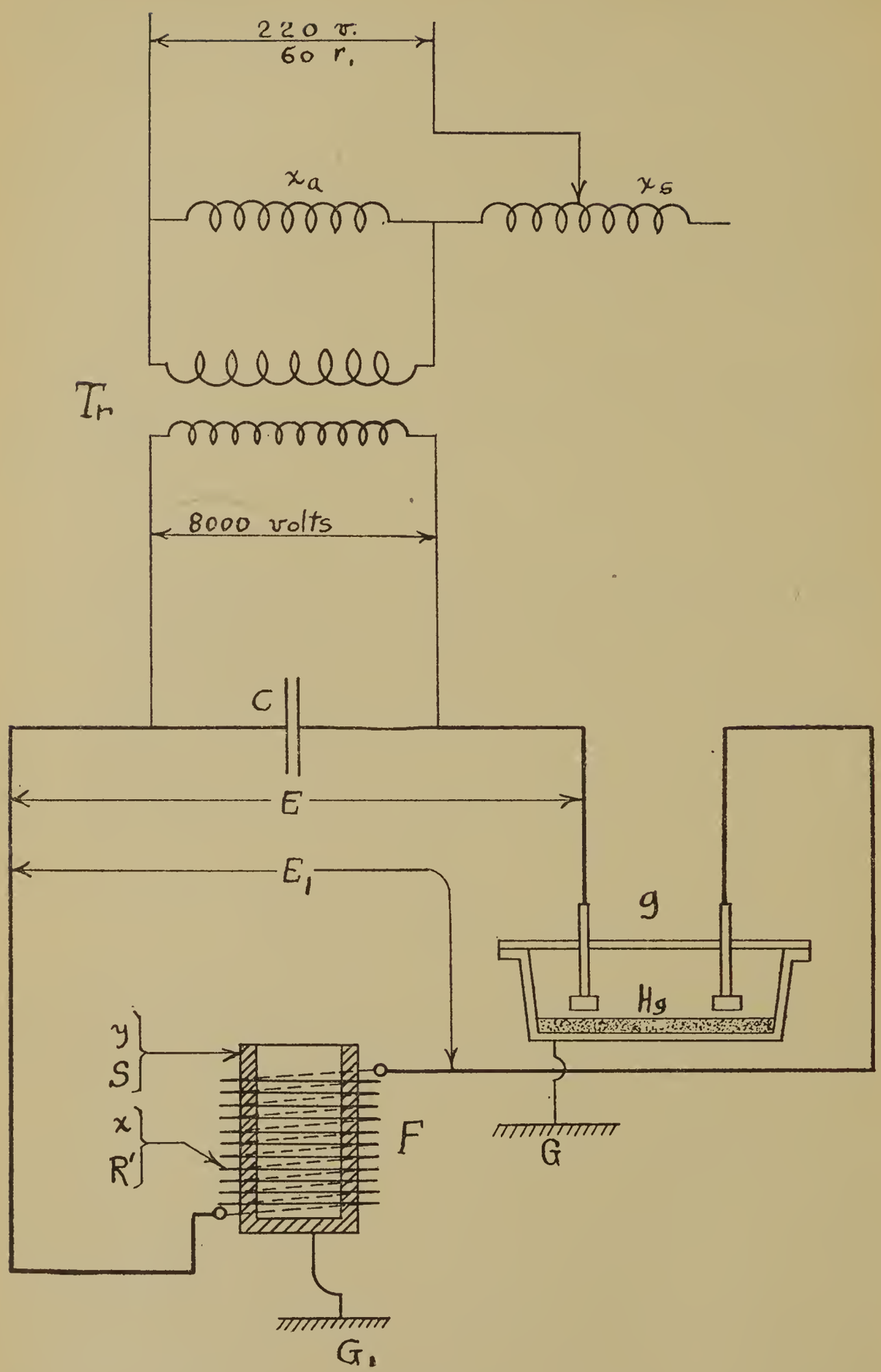


FIG. 20.

power factor has been developed by the writer but its value has not yet been satisfactorily ascertained by experiment. It may or may not be used. It may be mentioned that the theoretically good results which may be obtained by the use of a shunt reactance, as x_a, in the matter of phase control and improvement of power factor, have so far turned out upon investigation to be somewhat disappointing. It is probable that when the gap ruptures a large number of higher harmonics are introduced into the current wave which have no power component. Thus while one is perfectly able, by means of a shunt reactance, as x_a, to make the fundamental current wave either lead or lag the impressed e.m.f. it does not seem possible to make the power factor (defined as the true watts divided by the volt amperes) exactly unity.

A power factor of 50 percent is easily obtained and with a little care in the adjustments the power factor may be raised, under favorable conditions, up to 60 or even 75 percent. This appears to be, at the present time, about as high a power factor in large power units as can be obtained when a ruptured gap is used for the production of oscillatory currents.

The function of the reactance x_s is to limit the amount of power supplied to the oscillatory current system. By suitably varying x_s the power input may be varied through a wide range without waste of power, as occurs when rheostats are used for the control of power input. The proportioning of the various parts of the system of circuits to meet various conditions may be satisfactorily determined by any one who has a full understanding of the mathematical theories and equations which have been given above.

Experiments have shown, that good results may be obtained from a supply source of 60 cycles and either 110 or 220 volts. The secondary e.m.f. of the transformer should not be less than 6,000 volts and may be, with advantage, 8,000 volts, if the condensers are made of one-eighth inch thick window glass, and the shunt reactance x_a is used; 6,000 volts is sufficient when x_a is not used.

As previously stated (p. 119), the effective value of the voltage E_1 will be much less than the effective value of the voltage E. The peak value, however, of the voltage E_1 may be nearly that of the peak value of the voltage E, and it is the peak value which

determines electrical breakdown of insulation. This fact must be considered in designing the insulation between turns of the inductor coil and between the inductor coil and the crucible. If the insulation between the inductor coil and the crucible is suffi-

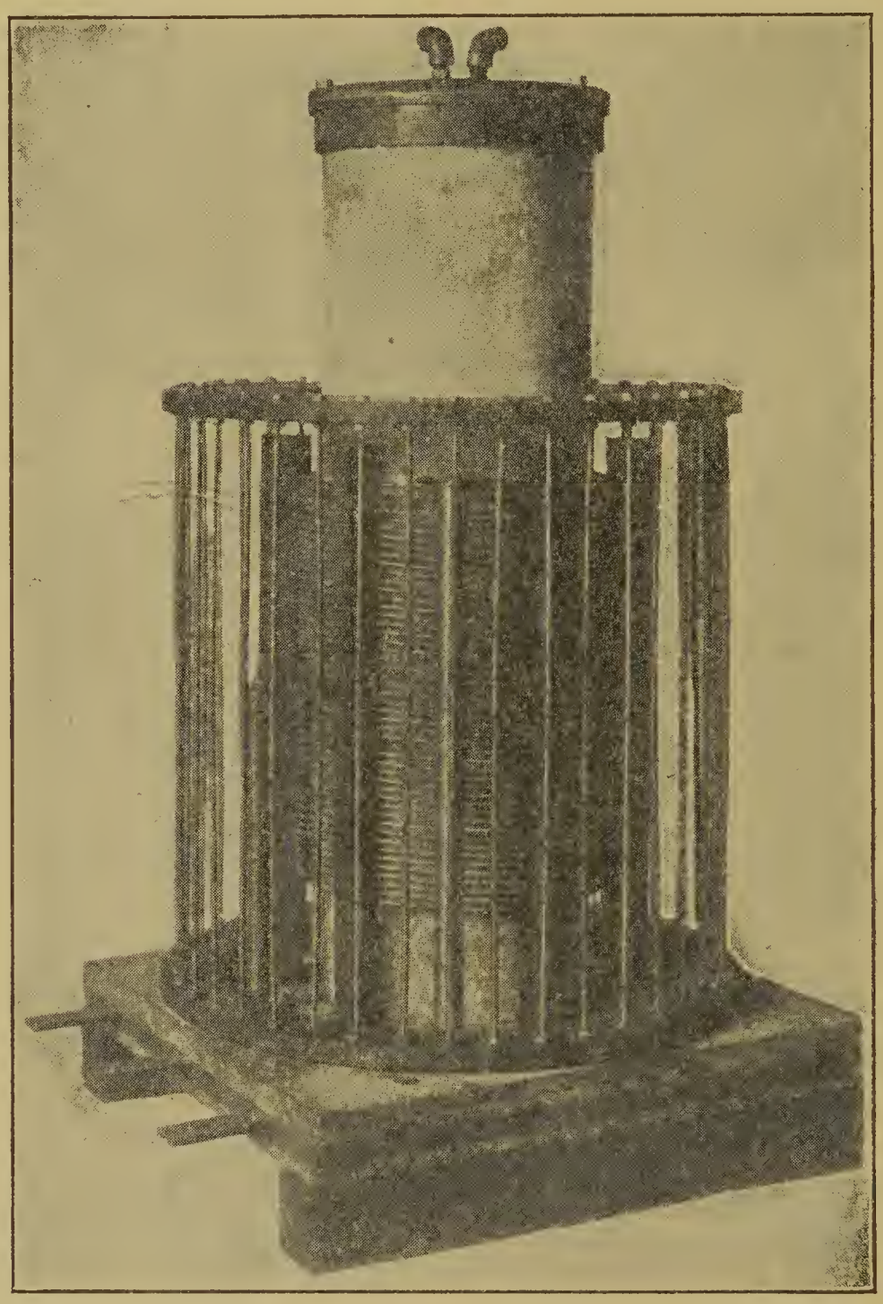


Fig. 21.

ciently high, then both the crucible and the metal pot of the discharge gap may be grounded as indicated at G_1 and G. In this case both the crucible and the mercury adjustor of the discharge gap may be handled with impunity, when the system is operating.

Fig. 21 is a photographic reproduction of a furnace suitable

for making melts in vacuum. This corresponds to the part of the diagram designated F in Fig. 20. Its inductor coil has 52 turns and an outside diameter of 19 cm. (7.5″). As the peak voltage applied to this coil may be as high as 7,000 or 8,000 volts, a quartz cylinder is used to electrically insulate the inductor coil from the crucible which is on the inside of the quartz cylinder. The crucible is 14 cm. (about 5.5″) in its outside diameter. The space between the crucible and the quartz cylinder is filled with some high-melting, good heat insulator, as "silox." As there are 52 turns in the inductor coil there may at times be as much as 150 to 200 volts between any two turns. This requires that the turns shall be separated from each other by 1.5 to 2 mm. Hence, to get as many windings as possible in a given length and give to the conductor sufficient cross-section to carry the current and to keep its resistance per turn low, use is made of edgewise wound flat copper-strip for the inductor coil.

IX. Circuits Used for Heating with High Tension Two-Phase Oscillatory Discharge.

In Fig. 22 are shown the complete connections employed for heating a single crucible using both phase A and phase B of a two-phase supply.

W_a and W_b are integrating watt-hour meters and A_a and A_b are indicating ampere meters. It will be noted that the discharge gap has three electrodes, a, c, and b. The inductor coil has a tap at its middle point C which connects to the center electrode c of the discharge gap. The free ends of the coil join one to the condenser C_a in the A phase and one to the condenser C_b in the B phase.

As two-phase operation would not in general be employed unless a large amount of power is being used, it is desirable, though not essential, to use the shunt reactances x_a and x_b. The other parts of the circuits are made clear by the diagram.

The power input by each phase may be separately adjusted by the reactances x_s and x_q. The phase of the currents I_a and I_b is theoretically under control by means of the shunt reactances x_a and x_b. These reactances remain fixed in value as long as the capacities, C_a and C_b remain fixed. They need not, therefore, be made variable, but should be chosen of the right value, once for all.

The pot of the discharge gap and the crucible may be grounded, as in the case of single-phase operation.

A large number of experiments were made to determine if any

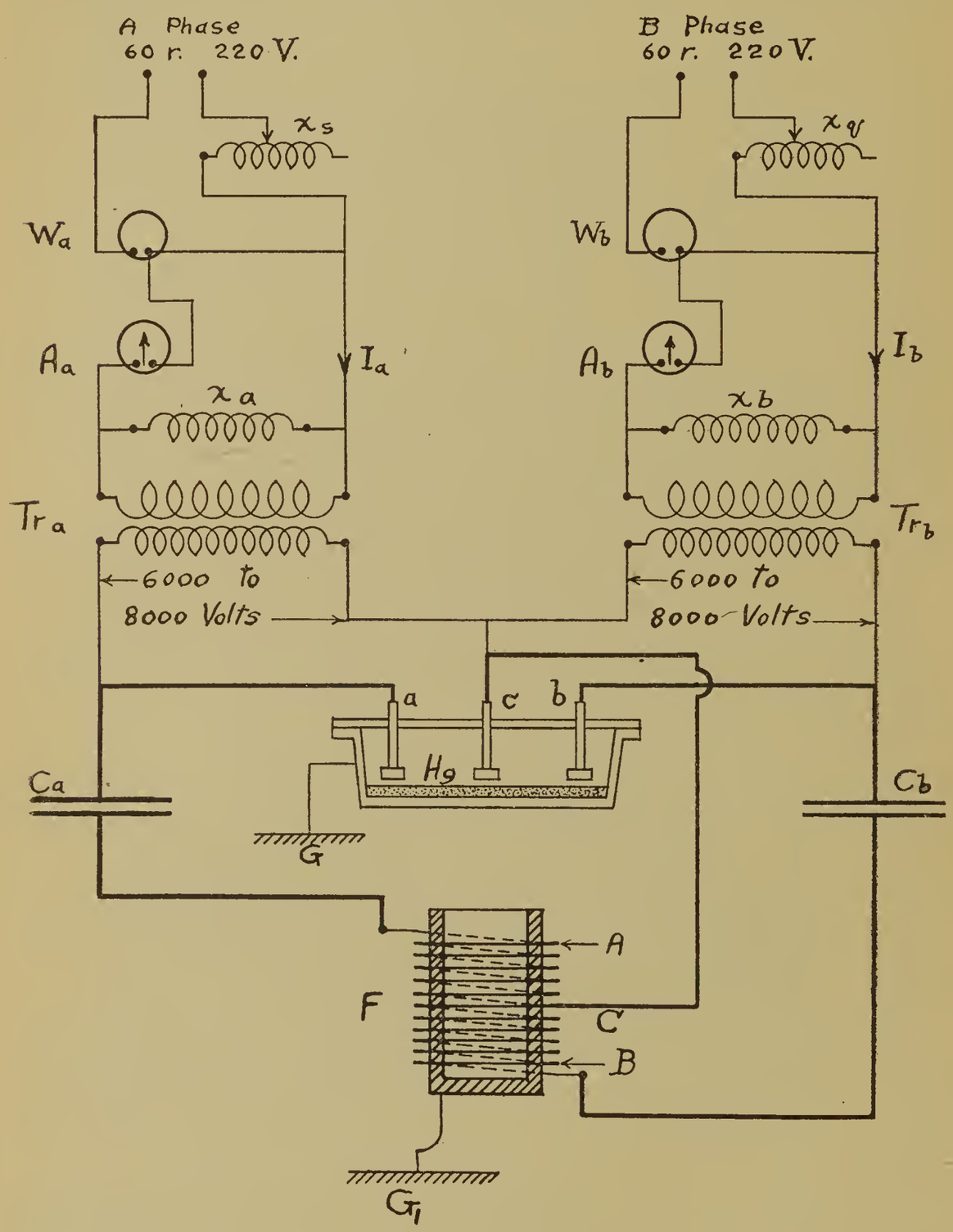


FIG. 22.

modifications in these circuits would give equally satisfactory or better results and the conclusion reached is that the circuits as shown in Fig. 22 are the best possible.

In Fig. 23 is shown a photographic reproduction of a two-phase oscillatory current system. There are 20 condenser units in the *A* phase and 20 in the *B* phase. Thirty kilowatts may be obtained from each phase, hence this constitutes an oscillatory current system for a 60-kilowatt furnace.

In Fig. 24 is shown a photographic reproduction of the transformers and switchboard. The two shunt reactances are not shown, as they were not in use when the photograph was taken.

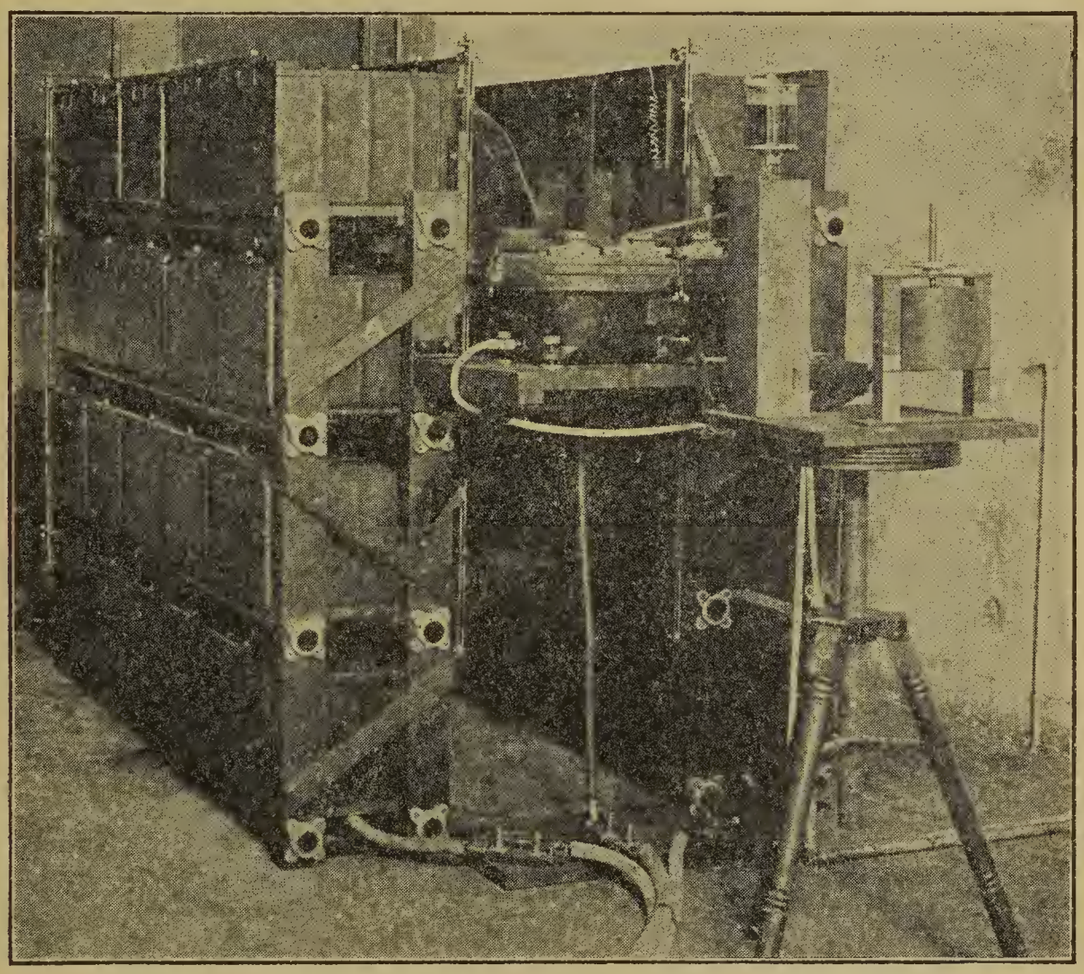


FIG. 23.

In Fig. 25 is shown a photographic reproduction of one form of furnace used with the equipment shown in Figs. 23 and 24.

In Fig. 26 is shown the same furnace with part of its outer casing removed to expose its construction.

PRINCIPLE OF THE "FOCUS INDUCTOR."

The focus inductor is a device which makes it possible to very closely couple the inductor coil with the mass to be heated and

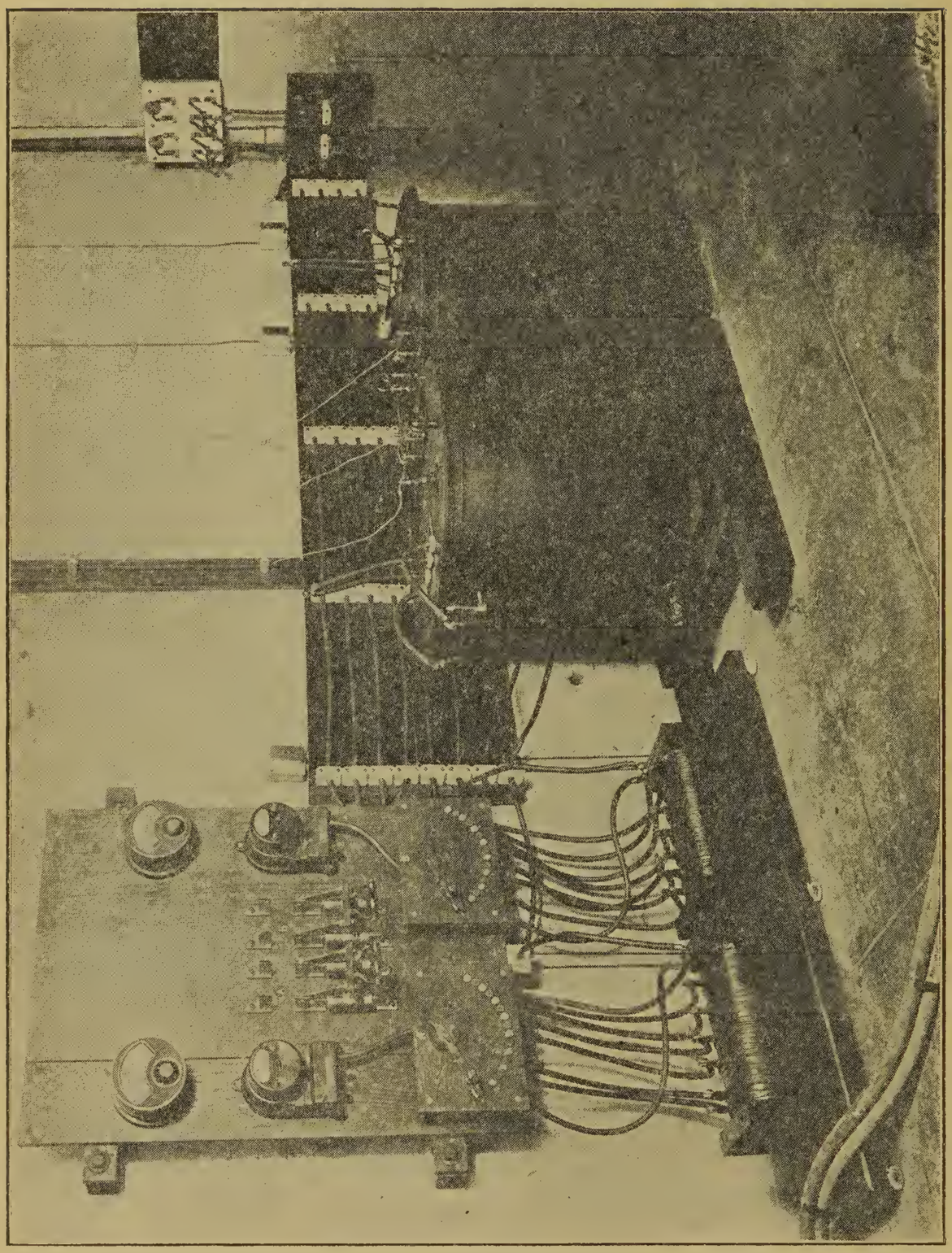


FIG. 24.

at the same time to greatly improve the conditions for effective electrical insulation when the heated mass is grounded. It permits the exposed parts of the furnace to be operated at low voltage and thus makes the operation of the furnace absolutely safe though many thousand volts are applied at the terminals of the primary inductor coil.

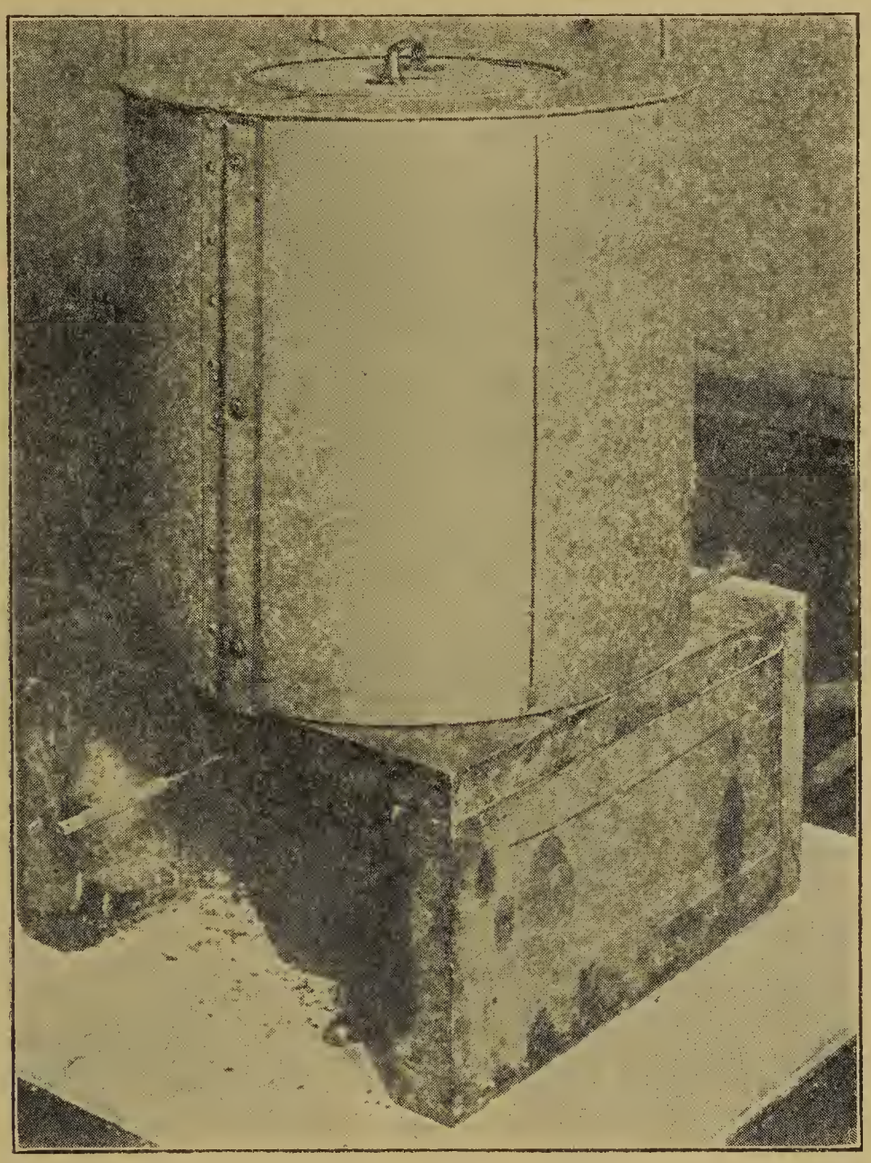


Fig. 25.

The arrangement to which we have applied the term "focus inductor" is indicated diagrammatically in Fig. 27.

In Fig. 28 is reproduced a photographic view of one form of focus inductor designed for rapidly heating a small mass to a high temperature.

In Fig. 27 *C* is a primary inductor coil, generally of 50 or more

turns. Within this coil is closely fitted a micanite cylinder (micanite of special construction made free of organic binding material) to electrically insulate the primary inductor coil from the focus inductor *f*. The focus inductor has one turn only, though it may consist of many straps of copper or other conducting metal, bent into the shape shown in the figure, the different

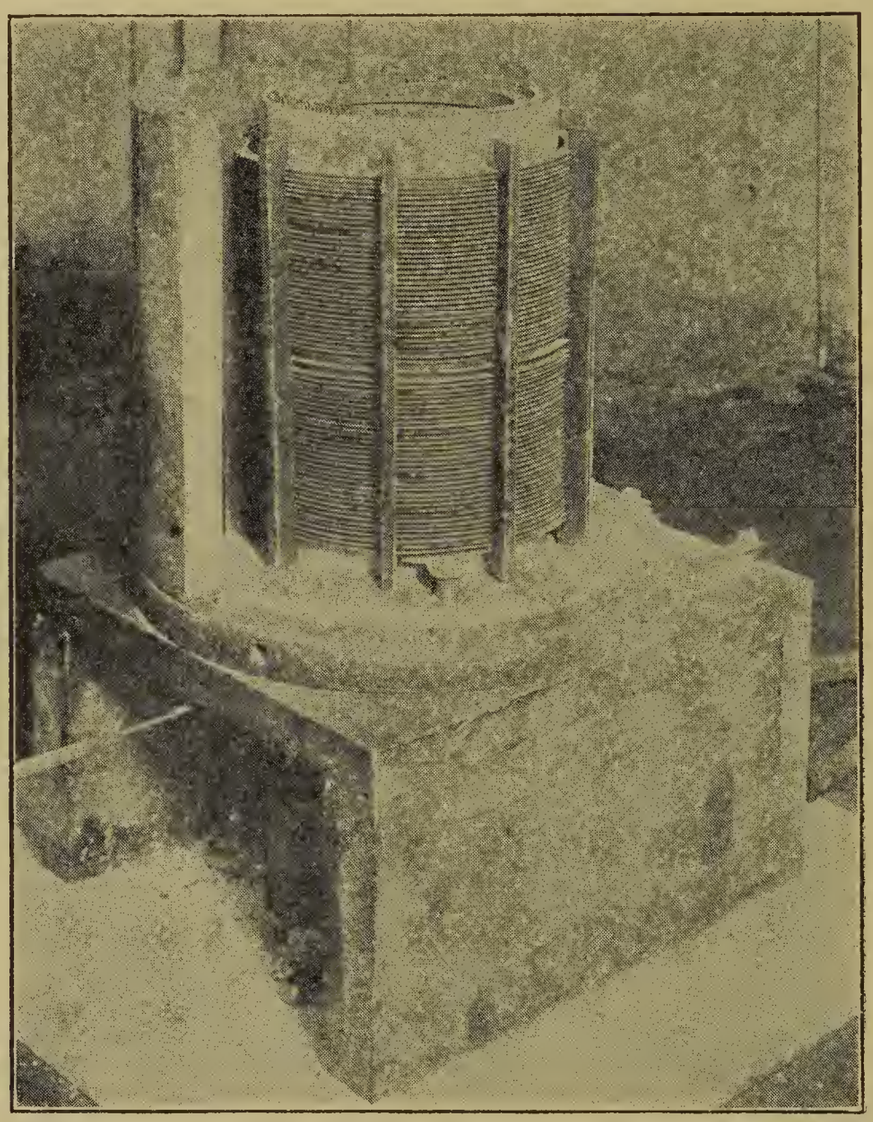


Fig. 26.

straps being placed one above the other until a sufficient number are used to build up a structure equal in its axial direction to the length of the primary inductor coil.

The interior portion, *p*, of the focus inductor, incloses the crucible or mass to be heated. If the temperature to be attained is not to exceed 1,300° C., the portion *p* of the focus inductor may be made of michrome or monel metal (this latter melts at

1,370° C.) and the mass to be heated may be chosen of a size to fit this portion closely. In this case the primary inductor coil has in effect almost unity coupling with the mass to be heated. Heat insulation may be used in the space K exterior to the portion p of the focus inductor, or, if less close coupling is required, heat insulation may be used only in the space h around the crucible.

The crucible and focus inductor can come into contact and be grounded at the point q.

When a high-frequency e.m.f. E (which may be at high tension) is applied at the terminals t_1, t_2 of the primary inductor

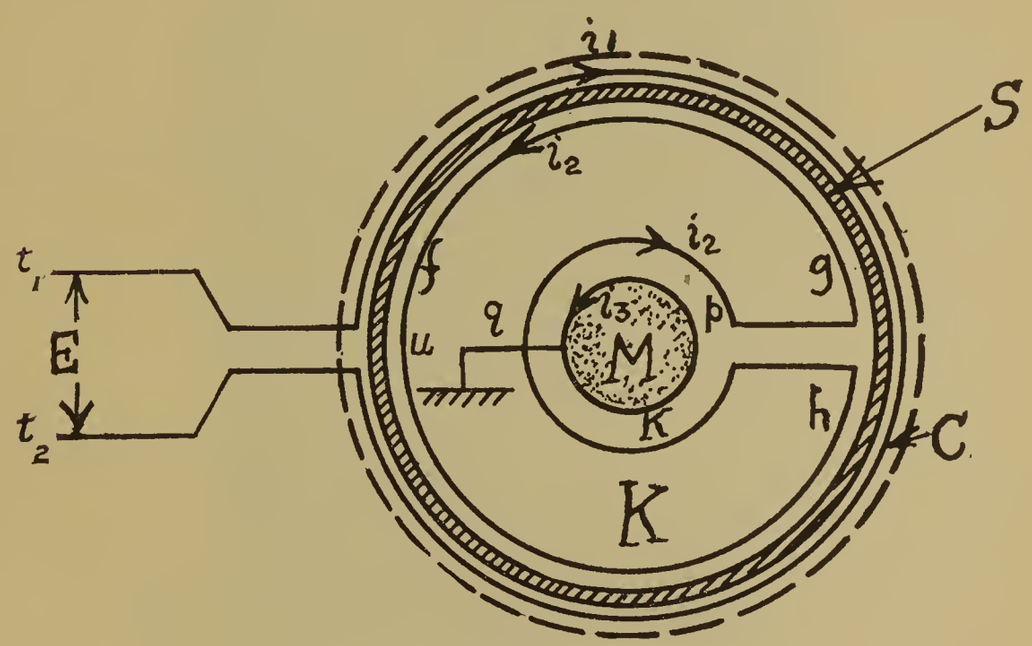


Fig. 27.

coil, currents will circulate in the primary inductor coil, the focus inductor and the mass being heated, in directions which are related to one another as indicated by the arrow heads in Fig. 27.

Since the focus inductor consists of one turn only, it will carry approximately N times the amperes which flow in the primary inductor coil when this coil has N turns.

It is easy to see that the introduction of the focus inductor into the primary inductor has the effect of focusing the electro-magnetic field upon the mass to be heated, and the only power which its use causes to be wasted is the I^2R heat loss due to the currents which circulate round it. This loss is small and is many times compensated for by the increased closeness of coupling gained.

The insulation S is never brought to a high temperature and consequently any desired e.m.f. may be applied at the terminals of the primary inductor without danger of electrical breakdown.

The primary inductor coil may be surrounded, as indicated in dotted line, by a grounded wire cage. Thus all parts of a furnace which may be at high tension lie between an inner and an outer grounded metallic enclosure. The operator is thus effec-

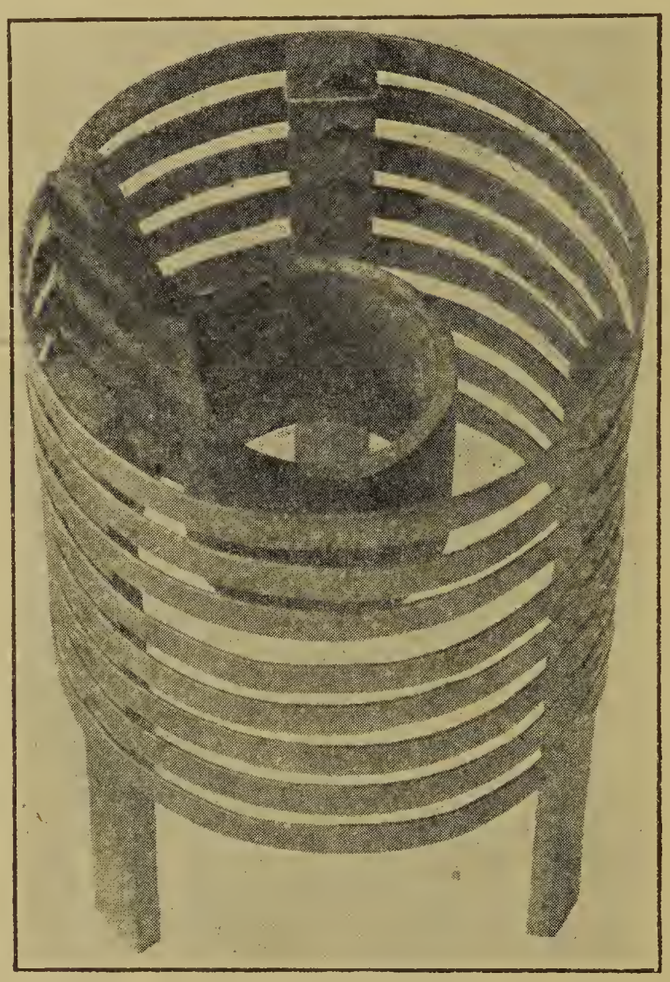


Fig. 28.

tually guarded from contact with all high-tension parts, and no electrical insulation against high tension is required where the temperature is high.

The focus inductor has been successfully and efficiently employed in a 25 K.W. single-phase furnace and in a 50 K.W. two-phase furnace designed for melting the non-ferrous metals.

A Dixon or Bartley graphite-clay crucible may be used to hold the melt, because these crucibles become somewhat conducting when once hot and the melt itself is also a suitable conductor.

When a focus inductor is employed in connection with a brass-melting furnace, the portion *g-q-h* is constructed of monel metal straps and the portion *g-u-h* of copper straps. If the crucible of poor conductivity is empty the monel metal soon heats to about 1,100° C. and the crucible then becomes heated by heat conducted to it from the monel metal. When the crucible is once hot and filled with scrap metal heating proceeds thereafter by induction.

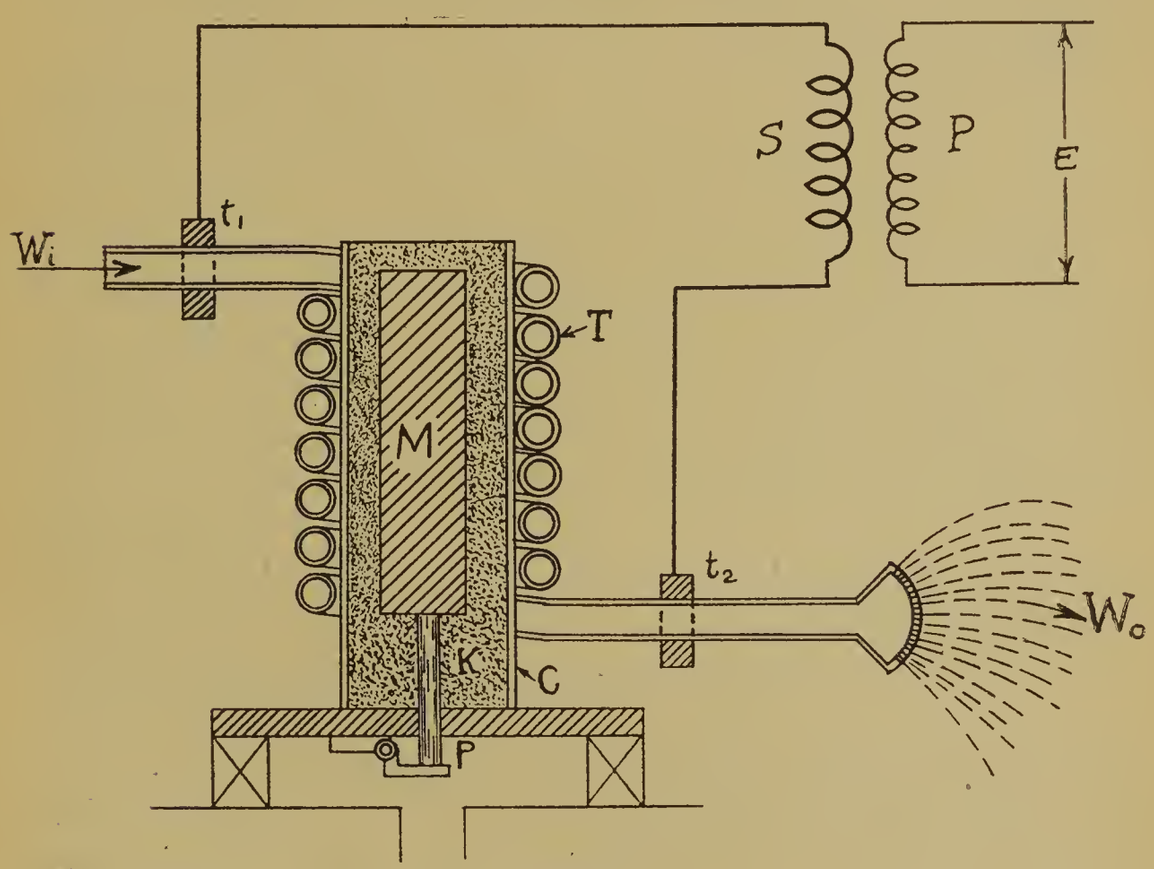


FIG. 29.

ARTIFICIALLY COOLED INDUCTOR FURNACE.

It is a frequent requirement in the melting of the ferrous metals and their alloys, to be able to obtain a very high temperature, 1,800° C., or over, and to preserve the melt absolutely free from carbon or other elements which, even in a slight degree, combine with the melt and so affect the chemical purity.

These ends are easily obtained by means of the arrangement shown diagrammatically in the cross-section view, Fig. 29. Here *C* is a hollow cylinder of quartz, micanite or like material. *T* is a copper tube of ten or more turns wound around the cylinder.

Provision is made for circulating water through this tube, the water entering at W_i and leaving at W_o as a spray (to prevent an electrical short-circuit of the tubular copper coil by the conducting water).

The ends of the copper tube t_1, t_2 may be joined to the secondary terminals of a suitably designed air transformer. A high-frequency e.m.f. (10,000 cycles or more) is applied to the primary of the transformer. Any conducting, cylindrical mass M located within the tubular copper coil has powerful currents induced in it, if the power supplied from the secondary of the transformer is sufficient.

By embedding the mass M in fine silica sand, CaO, MgO, Al_2O_3, zirconia, or like highly refractory non-contaminating material, the mass becomes sufficiently heat-insulated so that the rate at which heat is acquired much exceeds the rate at which heat is lost. The circulating water maintains the inductor tubular coil cool and its resistance low, regardless of the power supplied to it. The temperature of the mass M rises rapidly, and, in the case of iron, melting occurs in a few minutes, the time depending of course on many circumstances, as: the power supplied, the closeness of the coupling, the quality and thickness of the heat insulator, the frequency of the current, the design and ratio of the air transformer, etc.

When fusion is complete superheat comes rapidly, the temperature rise becoming accelerated because the resistivity of the cylinder when molten much exceeds its resistivity previous to fusion. The molten metal retains the form it had before fusion, lying in the powdered refractory as molten iron lies in its sand mold.

Since the inductor coil is maintained cold by the water circulation, a temperature gradient from the melt to the coil of 1,800° C. or over may be maintained in the distance of a centimeter or less. There appears to be no theoretical upper limit if the refractories will endure, to the carbon-free temperatures which may be obtained in this way. There are no crucible troubles to meet, because the pocket in the powdered refractory serves as crucible.

When fine sand is used as the refractory a thin wall of silica fuses about the molten steel and entirely prevents any outward flow or leak from the molten mass.

When the melt is complete the molten metal may be drawn off by removing a plug *p*, or the metal may be allowed to freeze in position and then lifted out as a solid cylinder.

In carrying out this process it is not essential that the mass *M* be a solid cylinder at the beginning of the operation. Small broken masses of metal may be placed in a molded cylinder of suitable material and thin wall and this be placed in position in the inductor and filled round with powdered refractory. The writer has in this way many times melted both solid cylinders and cylinders formed of small pieces of iron and then raised the molten mass to an intensely white heat, well above the melting temperature of platinum.

This new method of obtaining carbon-free melts, very rapid heating and excessively high temperatures, it is believed, is perhaps the most useful and important application of high-frequency induction for the heat treatment of materials at high temperature. The quantities of materials which can be handled in this way seem only to be dependent upon the power which is available with currents which exceed 10,000 cycles in frequency, but its very promising applications in any extensive way still await investigation.

It is almost needless to remark that compressed air, or gases, may be allowed to flow through and expand in the tubular inductor to effect the cooling in place of water.

It should be remarked that when energy is fed very rapidly into a mass of molten material which lies in a non-conducting container, a very considerable circulation of the molten material is produced by the action of electro-magnetic forces. The molten material, if lying in a chamber of cylindrical form, flows upward along its axis and downward along its circumferential region. It is proposed to make important uses of this phenomenon for causing a thorough mixing of materials which are being heated, and other applications are now being studied.

CONCLUDING REMARKS.

The development of electric heating with high-frequency currents and without interlinkage of an iron magnetic circuit with an electric circuit, was brought by the writer to its present stage of development (December, 1918) almost entirely by laboratory

experimentation. In the course of this experimental work it has been proved beyond doubt that, using oscillatory currents to secure the requisite frequency, rapid heating of a crucible and its contents may be obtained with a thermal efficiency in the neighborhood of 60 percent and temperatures up to 2,200° C. or 2,600° C. may be obtained. Batches of several kilograms of such materials as Pyrex glass among the non-conductors and pure iron and pure nickel among the conductors have been rapidly melted in air and in vacuum. Platinum has recently been melted without the presence of carbon in a 20 K.W. furnace sold to the Baker Platinum Co.

After these experimental results had shown the correctness of the physical ideas which led to and guided the experimental developments, the precise nature of the problems involved became more and more clearly recognized. The necessity became apparent for developing in a comprehensive way a complete theory of this method of electric heating, and of other entirely new engineering matters which develop when high-frequency currents, in connection with large amounts of power, are utilized.

These theoretical foundations have now been laid and all the basic mathematical relations have been found which are necessary for, and will permit the design of, very large power heating units.

For electric furnaces which require the use of 100, 500, 1,000 kilowatts or more, it is highly probable that the best and most practical source of the high-frequency current required is, in the light of present practice, an electric generator direct connected to a steam turbine.

Practical considerations demand that the voltages applied to, and used near, the heater unit shall be well below any voltage which might be dangerous to life. The mathematical treatment given above proves that there is no theoretical limitation in this method of heating, either to the voltages which may be safely used or to the size of the units and the amount of power which may be employed. Theory also shows that in all cases where the high-frequency current is obtained from an alternator the supply lines can furnish power at practically unity power factor and that the possibilities of obtaining high thermal efficiencies and rapid heating and elevated temperatures are equal to or better than the

results obtained in these respects by standard methods of electric heating. Furthermore, both theory and experiments have proved that uniformity in the temperature of the heated mass, perfection in the control of temperature and the ability to heat up charges from the cold with great rapidity exceed anything heretofore obtained in electric furnace practice.

The path is now laid and the mathematical formulae are given for the purely engineering advance in the designing of large power units for heat treatment of materials (glass, non-ferrous and ferrous metals) in tonnage quantities.

The oscillatory current system for obtaining the required current of high frequency has not so far been utilized in power units exceeding 60 kilowatts. The large expense of the condensers required, which are only active in the oscillatory current method from one-fifth to one-sixth of the time, the rather low power factor connected with this system and the high voltages required for generating the oscillations, would seem to limit this method of obtaining high-frequency current to single power units of the order of magnitude of 100 K.W.

Two or more oscillatory current systems may be made, however, to operate in conjunction to heat a single furnace of large dimensions.

Because, on the other hand, high-frequency currents may be obtained with oscillatory discharge with apparatus which contains no moving parts and which has an exceedingly slow depreciation, this system of heating is ideally adapted to small power installations and to all kinds of laboratory furnace practice.

The much larger problem, of the heat treatment of materials in tonnage lots, awaits for its practical solution the development of less expensive and more efficient means of obtaining currents of 10,000 cycles or more in limitless amounts. Intensive investigation is now in progress in several quarters looking to the development of high power, high-frequency apparatus, and full success in this matter will mean a great and highly satisfactory solution of the future commercial heat treatment of materials in large quantity by means of the electric furnace.

The presentation given above relates only to the basic principles which underlie this new method of electric heating and no attempt has been made to give data and results of tests upon the

performance of specific apparatus. A paper on this subject should follow later as the result of an engineering development which it is hoped the near future will see.*

Palmer Physical Laboratory,
Princeton University,
Princeton, N. J.

DISCUSSION.

SAUL DUSHMAN[1]: I would like to ask Professor Northrup whether very much can be accomplished with a kilowatt or two kilowatts of high-frequency energy in the melting of small amounts of metals or heating them to a very high temperature.

E. F. NORTHRUP: The best answer to your question is to describe a furnace which the speaker made for Prof. Hulett, of Princeton. He merits the distinction of possessing the smallest induction furnace in the world. The crucible is made of platinum and iridium alloy. It is 1 cm. in diameter and 1.5 cm. high. It is located in a quartz tube from which the air may be exhausted. A very small inductor coil surrounds the tube on the outside. A fraction of a kilowatt is sufficient to raise this crucible and its contents to near the melting point of the crucible. The temperature attainable in an induction furnace heated with high-frequency currents, is chiefly determined by the proportioning of the inductor coil and melt to the power available and to the character of the refractory material used as a heat insulator around the melt. Two kilowatts is quite sufficient to raise a crucible, which is only a few centimeters in diameter, above the temperature of melting platinum.

* Since the presentation of the above paper a development into commercial form of high-frequency induction furnaces of the oscillatory current type has been actively continued. Many installations are in service. Further tests were made of the principles given above, employing a high-frequency alternator as a source of high-frequency current.

By courtesy of the General Electric Co. an Alexanderson high-frequency alternator was placed at the writer's disposal. This alternator had an output of 200 K. W. at 25,000 cycles per second. Copper in considerable quantities was melted at the rate of 8.38 lb. per kilowatt hour. With better designing of the furnace the pounds per kilowatt hour may be expected to increase considerably. In short, an efficiency of 80 percent, or slightly more, was demonstrated as entirely practical to obtain, while an efficiency of about 77.7 percent was actually reached.

Additional matter of interest on "Production of Carbon Free Alloys of High Melting Points" will be found in Bulletin No. 3, for April, 1919, published by the Pyrolectric Instrument Co., Trenton, N. J.

[1] Res. Laboratory, General Electric Co., Schenectady, N. Y.

Mr. Carrier: Mr. Northrup's remark about Jules Verne reminds me of some fearful and wonderful things I once read about Mr. Tesla's oscillator. Does Mr. Tesla's oscillator produce anything that would be useful in this furnace?

E. F. Northrup: Answering Mr. Carrier's remarks I might reply that I built a high-frequency coil at an expenditure of over one hundred dollars to satisfy certain persons who said: "Northrup, I wish you would make a high-frequency coil, so that we can have something to make a show with." So I entered a shop order to have one made for operation on an oscillatory current system yielding about 20 kilowatts. The voltage was stepped up to give a discharge of about 12 inches. You may be sure this discharge made some noise and fuss, but for producing results of value it is about the most useless piece of apparatus I know of.

Mr. Carrier: I might remark, in connection with Mr. Northrup's remark about the comparative safety of working with high frequencies, that I have turned on the current of an Alexanderson oscillator working at 20,000 cycles and I estimated that I got 10,000 volts, and all it did was to burn a nice, clean-cut hole through my coat all the way through, but it seems to me that Mr. Northrup's remarks about the advantage of high frequency and decreasing capacity are certainly such as to show that it would be well worth while considering higher frequency than 20,000 if we can get it.

E. F. Northrup: I do not think much will be gained in considering the use of extremely high frequencies for heating purposes. For inductive heating I think the optimum frequency will be found to lie between ten thousand and twenty thousand cycles per second. With higher frequencies than this, radiation losses and skin effects in the inductor coil play an important rôle. Fifteen thousand cycles per second is a very satisfactory frequency and one which may be obtained from high-frequency alternators constructed in large power units.

A paper presented before the Pittsburgh Section, February 14, 1919, and called up for discussion at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

THE ELECTRIC FURNACE: ITS INTRODUCTION INTO FOUNDRY PRACTICE.

By W. E. Moore.[1]

Abstract.

A discussion of the advantages of the electric furnace in the steel foundry, particularly in comparison with the open-hearth furnace and the small Bessemer converter. The author concludes that it is superior to both of these, in regard to cheapness of raw material (the electric furnace can use a larger proportion of small scrap), conservation of alloying metals, waste of lining, temperature obtainable, control of chemical composition, and quality of steel produced. He prefers the acid-lined electric furnace because it turns out somewhat better steel and castings.—[J. W. R.]

It is pretty generally known in metallurgical circles that the electric furnace produces a grade of steel product superior to other standard processes. It is also pretty generally understood that the electric furnace is specially adapted for making fine tool steels and alloy steels, for with the basic-lined electric furnace it is quite possible to reduce the sulphur and phosphorus contents of steel to any degree or commercial limit required, as for instance to 0.02 percent.

On account of the possibility of maintaining a reducing atmosphere and a carbide or reducing slag in the electric furnace, it is more feasible to finish the steel to what is known as a "dead melt" or to thoroughly "kill it," than by any other process. When steel is thoroughly "killed" and held for some time under a reducing slag, it then becomes entirely feasible to alloy it with the various alloys, such as ferro-manganese, ferro-chromium, ferro-uranium,

[1] President, W. E. Moore & Company, Engineers, Pittsburgh, Pa.

ferro-vanadium, etc., with negligible loss or no loss at all of the alloying metals; whereas with the open-hearth furnace there is generally a very large loss of the alloys, which are often quite expensive. Thus the ability of the electric furnace to produce steel of low S and P content and to make it very solid by degasifying, or "killing" with the carbide slag, together with the accuracy to which the chemistry may be carried out, and the fact that its product may be alloyed without loss, make the electric furnace the most efficient means of manufacturing high-grade alloy and tool steels.

It is, however, not so well known that the electric furnace has lately become the most economical means for producing suitable electric steel for castings wherever power is available. The electric furnace is by far the most rapid, accurate and clean method for producing any grade of steel. In steel foundry practice speed is specially important, for with the heats coming so rapidly, one after another, a smaller furnace can be used for a given foundry capacity, thus enabling the required tonnage of castings to be produced with a lighter crane, ladle and building; less floor room and a smaller set-up of moulds. By the same token, the steel can be poured off more rapidly, so that there is less loss by skulling of the ladles and by "miss-run" and "cold-shut" castings, thus increasing the yield.

As is well known, the open-hearth furnace will produce excellent steel for castings, but the temperature which it is practicable to obtain is always low compared with the electric furnace, even for heavy thick castings by about 200 to 300° F. (100-150° C.), practically limiting the open-hearth process to heavy castings.

With the electric furnace it is easy to superheat the steel to any degree desired without any "hit or miss" rule, as with the converter, so that small thin castings, similar to iron castings, are commercially possible.

Heretofore, the side-blow converter has been the most feasible means of producing steel for small steel castings, and has replaced the crucible process for such work; for, with the side-blow converter, if carefully operated, and after the first two heats in the morning, the steel can be made quite hot and fluid. The process of blowing the steel is, however, very objectionable, in that it fills the steel full of oxides and gases and makes it quite porous. In

order to keep the heats up in the converter, it is essential to use pig metal of high silicon content, for which an extra price must be paid, or else ferro-silicon must be charged, which is still more expensive. Metal losses in the converter process are quite high, ranging usually from 16 to 24 percent, and as no refining whatever is done, these heavy losses tend to concentrate the impurities that were in the metal charged, so as to still further contaminate the blown steel. Therefore, it is not the practice to use the cheaper grades of scrap, but high-grade and more than ordinarily expensive metals must be charged. The steel is so filled with oxides and gases that an extra quantity of ferro-alloys must be added in order to "kill" the steel; frequently two times or more the quantity of alloys are required for converter steel as are necessary with electric steel.

The converters are silica lined, and as the oxides of the steel are produced in quantities, forming a strong flux for silica, the fluxing away and erosion of the silica linings is extremely rapid, so that in the converter process practically twice the expense for refractories is involved as is required in the electric furnace process. Labor is also greatly economized by the electric process.

With the electric furnace, since the furnace atmosphere may be readily maintained in a reducing condition, no oxidization of the charge takes place during melting and it is possible to melt thin, light scrap, such as iron turnings, which in the converter process or in the open-hearth process would be oxidized to such a large extent that it is practicable to melt only a comparatively small percentage of the total charge of such scrap. For the electric furnace, such scrap as "machine-shop turnings" ordinarily sell at about $6.00 to $12.00 below the market price of heavy melting scrap, which is the grade required by the open-hearth process. With the open-hearth process it is also essential to melt a large percentage, frequently 50 percent, of pig, which usually costs a still higher price than heavy melting scrap. The modern-type high-power electric furnace operates at a power-factor ranging from 85 to 90 percent, and is usually sold complete with sub-station equipment to receive high-voltage power.

The melting losses in the electric furnace are far less than with any other process, usually running about ½ to ⅓ of the loss in the open-hearth process and about ⅙ to ⅛ of the loss in the converter process.

In the ordinary steel foundry, making small medium and light weight castings, the most favored size of electric furnace has a nominal capacity of 3 tons per heat, though sometimes the smaller foundries call for the 1½-ton size. There is seldom anything larger than 3 tons required, for a properly constructed electric furnace can, by having its banks built up, be used to produce 50 percent larger than rated heats. Under favorable conditions a 3-ton furnace will make 12 heats of steel per 24 hours, and use about 550 K.W.H. per ton of liquid metal starting with cold scrap, and melting, refining and superheating the metal, suitable for foundry purposes. A 3-ton rapid-type electric furnace has a record of producing steel sufficient to make and ship over 1,000 tons of light-weight castings running 100 pounds and less in weight, in one month. Assuming a yield of 66 percent, which is good practice for such light castings, such a furnace would turn out over 60 tons of liquid steel per day.

Such a furnace would ordinarily use 18 to 20 pounds (8 to 9 kg.) of carbon electrodes per ton of steel, costing now 9 cents per pound (before the war 4½ cents) and at a refractory cost of some 30 to 40 cents per ton.

For foundry practice, both basic- and acid-lined furnaces are used. When the basic furnace is used, no great amount of refining is carried out, and no slag is taken off, for at present the most exacting specification calls for S and P limits of 0.05 percent and under, as distinguished from S and P limits of 0.02 percent in tool steels. With the basic furnace, any portion of the basic slag entering with the steel into the molds, which are necessarily silica lined, tends to produce surface blow-holes, and consequently, with basic-lined furnaces, it is customary to use only bottom-pour ladles, which are more expensive to maintain and slower in pouring small castings.

Largely for these reasons and for the simpler technique involved in the operation, the acid-lined electric furnace is generally preferred for foundry work, as low cost scrap, being obtainable of suitable grade for melting to the required S and P limits, is readily obtained.

With the electric furnace, steel castings of medium and light weights are now ordinarily made in green-sand moulds faced with

silica sand, and gated very much as in iron-foundry practice. It is, however, necessary with all steel castings, to add "sink-heads" or "feeders," to take care of the extra shrinkage.

The electric furnace, when substituted for converter furnaces, has been known to increase the yield of castings in percent of metal charged to weight of castings shipped, from an average of 40 to 45 percent with the converter to 60 to 70 percent with the electric furnace, when on such light castings under identical conditions.

Such furnaces enable the steel foundry to make almost any thin light casting, of practically any chemical composition or degree of alloying required, and usually effect a saving of some $30.00 per ton over the converter process and a somewhat smaller saving over the open-hearth process. As these advantages of the electric process are becoming known, the electric furnace is rapidly replacing converters and is beginning to replace the open-hearth furnace in foundries.

The crucible process for foundry steel is now practically obsolete, having first been replaced by the converter process, which in turn is being replaced by the electric process.

The electric-furnace steel, when made to the same chemical specification, is always better than other steel. Usually such steel will run 15 percent or better in tensile strength or in ductility, depending upon its heat treatment. It is easier to control the chemistry, and consequently, electric steel can be made to more accurate specifications than other steel. The following comparative physical properties of electric-furnace *vs.* open-hearth steel made to the American Electric Railway "Axle Steel Specifications" is shown by the tests of a prominent metallurigcal engineering firm for a large city street railway line. These steels, after forging, were held one hour at a temperature of 1,450 to 1,460° F. (785°-793° C.), then quenched in 65° oil, then drawn back at 1,185° F. (640° C.) and cooled slowly in the furnace.

	Open Hearth	Electric Steel
Elastic Lim. (lb. per sq. inch)	41,060	64,850
Tensile Strength (lb. per sq. in.)	89,100	105,140
Percent Elongation in 2 in.	21.5	22.0
Percent Reduction in Area	31.74	52.37
Character of fracture	Granular	Silky Cup
Elastic Torsion	16,750	33,700
Shearing Strength	62,400	76,000

The following is a typical operating cost (no overhead included) estimate, viz.:

Estimate of operating cost per ton of liquid steel, produced by Pittsburgh 3-ton rapid type electric foundry furnace, in continuous operation, melting heavy steel turnings and superheating the steel to a fluidity suitable for small steel castings.

Melting Stock.	
1 ton (2,060 pounds) heavy steel turnings—"Axle turnings"	$12.36
(3% (60 pounds) loss in melting—"Axle turnings")	
Electric power, 550 K. W. H., @ 1c.	5.50
Electrodes, Carbon, 20 pounds @ 9c. (Ante Bellum price 4½c.)	1.80
33 Pounds of Mill scale (used as a decarbonizer)	0.09
Additions.	
8 Pounds, 70% Ferro Manganese @ 10c.	0.80
5 Pounds, Ferro Silicon @ 5c.	0.25
(Note that Spiegel and 20% Ferro-Silicon may be substituted at lower cost).	
½ Pound Aluminum	0.17
Labor and Maintenance.	
Wear and tear on refractory roofs and linings @ 40c. per ton (some of our users average 20c. per ton)	0.40
Furnace Labor	1.00
Total Cost of Liquid Steel in the Ladle (per ton)	$22.37

COMPARATIVE PHYSICAL PROPERTIES OF ELECTRIC FURNACE *VS.* OPEN-HEARTH AXLE STEEL OF A. E. R. A. SPECIFICATIONS.

The following tests were made January 30, 1919, by R. W. Hunt & Company, Chicago, for the Chicago Surface Lines for the Chicago Electric Railway Company, and illustrate the physical properties of heat-treated electric-furnace steel; heated to 1,450-1,460° F. (785°-793° C.), held one hour, then quenched in 65° oil, and then drawn at 1,185-1,200° F. (640°-648° C.) for one hour, and cooled slowly in furnace.

ELECTRIC RAILWAY AXLE STEEL.

	A. E. R. A. Standard Specifications	Open Hearth L. S. Co.	Electric Steel G. S. Co.
Elastic Lim. (lb. per sq. inch)	55,000	41,060	64,850
Tens. Str. (lb. per sq. in.)	90,000	89,100	105,140
Percent Elongation in 2 in.	22	21.5	22.0
Percent Reduction of Area	45	31.74	52.37
Character of Fracture		Granular	Silky Cup
Elastic Torsion	.....	16,750	33,700
Shearing Strength	.....	62,400	76,000

DISCUSSION.

J. A. Coyle[1] (*Communicated*): I will refer first to the difficulty in charging an electric steel furnace, as that operation consumes from twenty-five to forty-five minutes, owing to the necessity of doing the work by hand. It is just possible, however, that a crane charging machine could be used advantageously, yet in our case we are limited to hand charging because of the clearance between the rail on our crane runway and the top of the mast of our furnace. I would say, however, generally speaking, that considerable thought should be given to the possibility of easily charging the electric furnace.

There is a second difficulty which occurs to me, and it is in the matter of slagging, that is to say, taking off the slag containing the phosphorus which has been removed from the steel. The generally existing oval hearth of electric steel furnaces does not permit of doing the slagging in a finished way, for the reason that it is difficult to withdraw all of the slag from the bath, particularly at a point between the pouring spout of door and the charging door, which results in some of the phosphorus going back into the steel when the temperature of the bath is raised to its highest heat. This would not be so serious were we dealing with steel to contain when finished only 0.030 phosphorus and sulphur at a maximum.

Another difficulty is that of the surge load of the electric steel furnace during the cold melting period. Some considerable thought should be given to minimizing the surge of current and the unbalancing of the electrical phases due to the surges which are characteristic of practice in electric steel furnaces of all kinds.

There is still further difficulty with electric steel furnaces which assumes serious proportions, in that the refractories forming the bottom, the side walls and the roofs of electric steel furnaces are not of sufficiently good quality to withstand the terrific temperature which it is possible to generate in electric steel practice, and on account of this fact there are excesses of solid non-metallic impurities reduced from the furnace walls and

[1] Universal Rolling Mill Co., Bridgeville, Pa.

the bottom which no amount of dead melting or killing will remove to the last degree; for, while the steel may appear to be sound, some of our more technical friends have found upon investigation microscopic evidences of these solid non-metallic impurities in the form of silicates and aluminates which detract from otherwise presumably good steel. This question of refractories suitable for electric steel furnaces is therefore one that should be given a great deal of attention and research.

I dispute the claim that the electric steel furnace is an active competitor of the open hearth furnace, nor is it reasonable to suppose that it should be. The increased temperature of electric steel produces a fluidity which is away beyond the fluidity of open hearth steel and where with open hearth steel it takes, say, one hundred and twenty-five pounds of fluid metal to make one hundred pounds of castings, with the electric steel furnace it takes one hundred and sixty-six pounds of fluid metal to make one hundred pounds of castings.

I recall that in making an investigation in Europe for Mr. Hess, of the Hess Steel Casting Company, the extreme fluidity of the metal caused a greater loss in sink-heads and risers than would have been caused in making open hearth tests.

Nor is the cost of lining per ton of the electric steel furnace at all comparable with the cost per ton with an open hearth furnace, in fact the lining cost in electric steel practice is a terrifically heavy and expensive thing, and, for this reason electric furnace steel cannot compete with open hearth steel for castings.

The advantage which is granted to the electric furnace is the possibility of reducing the phosphorus to a point where it cannot be found by chemistry, and the sulphur to mere traces, and the facility and ease with which this dephosphorization and desulphurization is performed; in our own practice it is not at all difficult to secure phosphorus and sulphur below 0.015 after having taken off one slag, and then finishing under the carbide slag.

It is not my opinion that the electric steel furnace exploiters are so much entitled to the credit for the performance of electric steel furnaces as are the metallurgists, the great electrical companies who manufacture the transformers and electrical control,

and the power companies. These three groups of industrial elements have done wonderful things in promoting the progress of electric steel furnace practice.

All that I have said does not necessarily mean that the electric steel furnace cannot produce good steel, as, on the contrary, it has produced some very magnificent steel, and to it belongs credit for much of the fine ballistic steels turned out during the war time.

I will relate to you some performances which have been carried out:

We produced about nine thousand tons of bullet-proof steel in thicknesses of 0.175 inch (4.4 mm.) and 0.225 inch (5.6 mm.). The first thickness was tested with a shot from a Springfield rifle at a range of fifty yards (46 m.) with a bullet leaving the rifle with a muzzle velocity of 2140 feet (624 m.) per second. The 0.225 inch (5.6 mm.) thick steel had to stop a bullet from a Springfield rifle at twenty yards (18 m.) range with the bullet leaving the muzzle at a velocity of 2700 feet (810 m.) per second. Both of these are very hard tests and I am happy to say that the yield in good plates from the large quantity turned out by means of the electric steel furnace was ninety-seven and one-half percent.

Again, in the matter of helmets, which were No. 20 gauge or 0.035 in. (0.9 mm.) thick, these had to stand a test with a Colt 0.44, with the bullet leaving its muzzle at a velocity of 840 feet (252 m.) per second, distance of range 10 feet (3 m.). The test had to be met without puncturing the helmet and I am happy to say out of one hundred thousand helmets made, not one failed in its performance.

In view of these facts, I can say that Mr. Moore is justified in his claims for the future performance of the electric steel furnace with regard to quality steel, but I am certain that the greatest good will not be forthcoming until such time as the exploiters and designers of electric furnaces get more actively and closely in touch with the metallurgists, with the great power companies, and with the electrical companies who manufacture the electrical instruments which are the brains of the furnace.

The future is undoubtedly filled with good prospects for a rapid and large advancement in the art of producing steel by means of the electric steel furnace.

W. E. Moore: With the old type electric furnaces, making basic steel, two to three or four heats a day is not unusual practice.

All of the costs are very much higher and are not to be compared with the much cheaper operation of the modern rapid type electric furnace, making acid steel.

It is evident from the analysis of the operation of the furnace plant under the care of Mr. Coyle, that he is operating with the old-fashioned, low capacity, slow type basic furnace. It is actually true that the rapid type electric furnace, when making acid steel for foundry purposes will under many conditions make it cheaper than the open hearth furnace.

J. W. Richards:[2] Do you wish to comment on the amount of steel necessary to make 100 pounds of castings?

W. E. Moore: Regarding the yield of castings, it is evident that Mr. Coyle has referred to some isolated operating conditions, where heavy castings were being made with the open hearth furnace, where the yield is always higher and has compared that to the output of some electric furnace plant, making much lighter castings; the yield is necessarily lower, when lighter castings are made, owing to the larger percentage of sprues and gates.

W. E. Moore (*Communicated*): Referring to the discussion by Mr. Coyle, I will answer his points in detail:

Time for Charging.—While 25 to 45 minutes may be required for charging a 6-ton furnace, where no conveniences are provided, I know of 3-ton foundry furnaces being charged to 4 tons per heat, with a charging time ranging from 11 to 13 minutes, and by using a charging machine, the charging time is reduced to 4 minutes.

Slagging.—The slagging operation with the old type electric furnace is slow and laborious, ordinarily three men being required for the operation, one on each side, working through the charging doors and another at the spout, to hook out the slag from the tap hole. When the slag is taken off through the back or slagging door, as with the Moore rapid electric furnace, where the opening is wide, only one man is required to do the work far more completely and with the wide door, the operator is better able to clean

[2] Prof. of Metallurgy, Lehigh University.

the slag completely out of the furnace, thus preventing re-absorption of phosphorus by the steel when under the carbide slag.

Surges.—It is a matter of great importance to the power company, in order to maintain its voltage without fluctuation and to the furnace user, in order to keep down his peaks which are measured for the "readiness to serve" charge by the power company, that the transformer be so connected as to distribute any unbalance in the load of individual electrodes and to have the reactance of the transformer properly adjusted to the local conditions to minimize the peaks and disturbance, due to sudden surges. This requires careful attention to the details of the power company's service line and generating equipment at the time the furnace plant is being designed.

Refractories.—With the exception of the electrodes, the refractories are the weakest link in the electric furnace construction and operation. While it is possible with acid furnaces to obtain roof lives of up to 350 heats, the life of the roof and other silica refractories in basic operation is materially reduced, due to the basic volatile matter from lime and other slag-making materials which is condensed upon the silica and gradually fluxes it down.

With basic operation, the life of the refractories may be readily reduced to from one-half to one-fourth of that obtainable with acid operation, even when making much hotter steel for foundries than it is customary to make with basic furnaces, when pouring ingots. A suitable basic refractory for side walls (above the slag line) and for roofs would greatly improve the operating conditions with basic electric furnaces. The extra costs for refractories and slags is one of the large items in the additional cost of basic over acid steel.

Electric vs. Open Hearth Steel.—It must be frankly admitted that the electric furnace under ordinary conditions is not a competitor of the open hearth furnace, where steel is to be made in large quantities, using hot metal as from blast furnaces. However, this story may be reversed, where power at reasonable price is obtainable and where the charge is to be made of cold scrap, as the electric furnace is capable of being operated with entire satisfaction on light, thin scrap, such as turnings, which are not

suitable for the open hearth furnace, which generally requires for its operation 50 percent pig iron. In such cases the cost of the scrap alone may run $6.00 to $12.00 cheaper per ton for the electric than for the open hearth furnace, which is ample margin to more than make the electric furnace competitive with the open hearth furnace in many situations.

For steel foundries, where extra hot metal must be had for small and medium castings and where the open hearth furnace fails by 200 to 300° F. to make the steel hot enough, the electric furnace is very much cheaper to operate and makes better and greater yield of commercial castings.

Yield in Weight of Castings Compared to Weight of Hot Metal.—It is true that the open hearth furnace in foundries is generally used only for very heavy castings and it is also true that with heavier castings, the yield, as a rule, runs much higher than with light castings. This would probably account for the yield mentioned by Mr. Coyle of 80 percent in the open hearth steel, as compared to 60 percent with electric steel. In fact, with very small castings, it is not possible to obtain as much as 60 percent yield, even with electric steel. Really, there is nothing in the theory that the electric steel being hotter, produces a smaller yield of merchantable castings. Indeed, the contrary is the usual result, for with light, thin castings, the hot steel made by the electric process gives a better yield, owing to the absence of cold shuts, and for heavier castings, where it is desired, the metal may be allowed to cool in the ladle, before teeming into the moulds, which has a tendency to produce somewhat better steel, since the non-metallic inclusions are thus allowed ample time to aggregate and float up to the surface in the ladle.

Comparative Cost of Electric Furnace Refractories vs. Open Hearth.—Comparing refractory costs, one must always consider whether the furnace is being operated basic or acid and, if basic, how many slags are being taken off. It is also important to consider the temperature of the steel, as to whether it is being superheated to a very high temperature for making light thin castings, or whether it is being heated only to a low temperature, as is usual for making ingots. Numbers of electric furnaces, designed and

built by the writer, in foundries making acid steel, are operating with a refractory cost of 40 cents per ton.

Reducing Sulphur and Phosphorus Content of Charge.—While dephosphorization can readily be produced in a basic open hearth furnace, as well as in the electric furnace, desulphurization to any large extent is not practicable in the open hearth furnace, for the higher heats become very destructive to the refractories, the fuel consumption goes up enormously and many complications arise in keeping the carbon content right, besides larger quantities of pig must be used.

With the electric furnace, the desulphurization is quite relatively cheaply performed, even to the very low limits mentioned of 0.015 percent sulphur.

Credit to Whom Credit is Due.—Like most engineering accomplishments, the electric furnace is not the result of a happy thought by any one inventor, but the modern rapid electric furnace is the result of systematic and combined engineering study and work, resulting from a specialized knowledge of electrical apparatus, metallurgy and of the science of power production and distribution. The furnace building company, which does not employ engineers having a good working knowledge of all of these conditions, cannot be expected to produce anything more than a hit and miss electric furnace installation.

Quality of Electric Steels.—In the experience of the writer, the electric furnace will not only produce steels of equal quality, but actually better quality than any other known process, the crucible not excepted. It can produce some grades of steel which are not possible in the crucible, such as steels of very low silicon content.

The entire conversion cost of electric steel is generally below even the cost of the pots in the crucible process, to say nothing of the very much more expensive high grade wrought iron charge necessary for the crucible process.

Chemical Control.—To make fine steels by the electric furnace process, as well as make good steels by any other process, it is essential to have the furnace operated under the direction of a competent metallurgist and with a properly equipped laboratory at hand. Indeed, in most installations for foundry steel, it is

cheaper to equip the plant with a laboratory, for economies can be effected in the way of buying cheaper scrap and meeting specifications for castings, where the services of a chemist are available.

The need for proper electric furnace regulators and a suitable equipment of instruments cannot be gainsaid, and it must be admitted that many electric furnace installations have suffered greatly on account of the inadequacies of instrument and regulator equipment, and the fact that the furnace installation has not been designed to work in harmony with the power system, upon which it is operated.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

IMPROVING THE QUALITY OF GRAY IRON BY THE ELECTRIC FURNACE.[1]

By George K. Elliott.[2]

Abstract.

Proposes to use a basic lined arc electric furnace to take gray iron from the cupola, to refine and superheat it, for special purposes. The cupola is strong on heating and melting, but weak in superheating, carbon regulation, waste of alloying metals, and impossibility of refining. The duplex process, using the electric furnace in tandem with the cupola, corrects and supplements the deficiencies of the cupola. It is also proposed to use it on direct metal from the mixer, to assure a superior quality of melted iron at a proper temperature for the foundry.—[J. W. R.]

The central facts and fancies embodied in this paper are either taken from or inspired by divers experiences with a basic-bottom Héroult electric furnace at the plant of the Lunkenheimer Company at Cincinnati, Ohio. Insomuch as these experiences extended over a period of nearly two years, and as the electric furnace is presented in a service different from usual, the author feels that his subject, at least, is justified by both a reasonably ample observation and a certain novelty of matter.

The aggression of the electric furnace into the field of steel castings has been widespread and irresistible; but, so far, the homely gray-iron casting has been beneath its exalted attention. Or, is it that the cupola furnace, venerable and efficient to a remarkable degree, has been a citadel too mighty to be overwhelmed by the invasion of the electric furnace, as were the established furnaces of the steel foundries? However it may be, the author

[1] Manuscript received March 3, 1919.

[2] Chief Chemist and Metallurgist, The Lunkenheimer Co., Cincinnati, Ohio.

believes the electric furnace has a very fair chance to break through into the domain of iron castings, at points in the line where the hammering of modern engineering for super-grades of castings is rapidly effecting a perceptible breach.

In regard to gray-iron castings, it cannot be denied that today there are classes of castings for which the insistent demand is for physical properties of a higher order than are commonly characteristic of such iron. Possibly every branch of engineering has some castings that should be, and sooner or later may have to be, made of better gray iron than is now used. To enumerate these instances *in extenso* is too great a task for this paper, and only a few will be given. One case, and the direct cause of the experiences here related, is that of steam valves for high-pressure and superheated steam. Both pressures and temperatures of the steam used in power plants have passed through a period of exceptional growth, placing upon the apparatus for handling steam severer demands than ever before. Similar is the case of cylinders for locomotives using superheated steam. Also there is the case of cylinders for internal combustion engines.

The two prime qualities most generally wanted in high-grade gray cast iron are strength and solidity, properties which usually are concurrent. The best strength gray iron shows is when under compression, and its worst, under impact and vibration; between come transverse strains and tension. If the cases are studied in which unusual strength of iron is required, it is discovered that almost always the castings are to be used where they will be subjected to either impact, vibration, or tension; or, in fact, where cast iron is at its greatest disadvantage.

Under the head of solidity may be included a number of related items such as density, closeness of grain, and freedom from subcutaneous imperfections such as slag inclusions, graphite segregations, blow-holes, shrink-holes, and the like. Solidity is an excellent goal for the aspiring foundryman; but, in going after high specific gravity he has to beware that he is not lured out from the realm of true gray iron into the dangerous domain of mottled and white irons where brittleness lurks to impair the impact strength of castings, and hardness to ruin their machinability.

Strong irons without exception require high pouring temperatures because they have high melting points and minimum fluidity. The irons most easily melted and, consequently, the most fluid ones, have high percentages of phosphorus in their make-up. Medium and high phosphorus irons comprise by far the greatest part of the melt of the gray-iron foundries of the country. They are the popular casting irons; and it is no extravagance to say that phosphorus has done as much or even more than any other element to popularize gray iron for castings. Still, phosphorus has the grave fault of unfavorably affecting the strength of iron, a thing it accomplishes by forming in the iron mass a network of structurally free phosphide which is quite brittle. Although easily fluid, high phosphorus irons are brittle and generally lacking in strength; conversely, tenacious, strong irons are necessarily low in phosphorus.

The great fact that strength and fluidity do not go hand in hand brings to light the greatest defect of the cupola furnace—its thermal limitations. It is the foundryman casting strong iron who knows best these limitations. The complete melting operation as it takes place in any furnace is divided into three obvious stages—preheating, melting proper, and superheating. For the first two, the cupola is the furnace pre-eminent; but for superheating it is not the same paragon of excellence.

All ordinary grades of gray iron—those with medium and high percentages of phosphorus—cannot be melted more easily or more economically than in the cupola. For melting efficiency this furnace takes precedence over its two closest rivals—the reverberatory or air-furnace, and the regenerative open-hearth furnace. For simplicity of construction, low cost of installation and upkeep, ease of operation, and economy of melting, it has no real competitor; and there is no probability that, for the great run of ordinary gray-iron castings, it ever will be displaced, although its improvement is conceivable. From the standpoint of process alone, the cupola is almost impeccable; but, from the high ground of product, it shows serious faults. The two worst of these are its weakness as a superheater for molten iron, and its incapacity as a refiner. The cupola is not a refining furnace and sometimes its hottest possible iron is not hot enough for the best casting results.

The thermal limitations of the cupola are inflexible and often tenacity has to pay the price of fluidity. If the demand for supertenacious and solid iron also becomes inflexible, there arises a dilemma which is capable of but one solution, and that is, the attainment of hotter iron. This can be done either through the elimination of the cupola with the substitution of a better superheating furnace, or through retaining the cupola and supplementing it with a more capable superheating furnace. In view of the unequaled economy of the cupola for preheating and melting, a method of supplementing is preferable. Supplementing means the introduction of a duplex process into the iron foundry.

Duplex processes are well established and of excellent repute among makers of steel; but, for the production of gray iron, they have been entirely neglected, leaving virgin a promising field for development in the making of molten iron for the highest grade of gray-iron castings. For a supplemental furnace to the cupola, there are several eligible types. Among these are the reverberatory furnace, the regenerative furnace, and the electric furnace. The air furnace, the author hardly considers sufficiently suited to the work. The regenerative furnace promises better, especially where a large tonnage is to be handled at one time. Still, in the end, the author believes the arc electric furnace presents the most alluring possibilities for really extraordinary castings.

The foundry with which the author has been connected has made a specialty of high-pressure steam-valve castings with extreme solidity exacted, and the tenor of his experiences there has been such as to prejudice him strongly in favor of the basic-bottom arc electric furnace as the best auxiliary for the cupola in a duplex process for the production of truly high-class gray-iron castings.

At this point we anticipate the question as to why the basic furnace is preferred. If superheating only is desired, it is entirely probable that the acid furnace should be given the preference; but if an important amount of refining is advantageous, the basic furnace should be used. It is true that the acid arc electric furnace exerts a considerable refining influence by virtue of the reducing condition so readily maintained in it. On the other hand, the refining tendency of the basic furnace is so much more pronounced and so readily responsive, that the iron charge

is perforce refined at the same time it is being superheated. By the time the charge has reached the pouring temperature it also has reached a highly refined condition; and, among other results, the sulphur has been reduced very materially, provided, of course, the proper basic and reducing slag has been maintained.

Sulphur in cast iron has so long been considered a necessary evil that its presence not only is commonly condoned, but occasionally is credited with certain benefits. No doubt there are a few of these last, for instance, where it aids in producing chill; but, in general, there is good reason to believe that iron is much better for its absence. That it induces unsoundness when in excess is general knowledge in foundries specializing in castings of a solidity that is a matter of severe test; high pressure valves are an example. Sluggishness of metal is a closely allied evil that travels in the train of sulphur and works unfavorably to solid castings. A medium or high sulphur content is inevitable in the product of the cupola furnace, while in the basic electric furnace its practical elimination is almost a matter of course. The basic electric furnace removes most of the sulphur while the metal is being superheated. This reaction makes possible for the electric furnace certain obvious compensating economies in the choice of raw materials used in the cupola phase of the process. Iron coming from the cupola with 0.100 percent sulphur, after undergoing about twenty-five minutes heating in the basic electric furnace under the proper reducing slag generally will contain about 0.030 percent. One heat the author has in mind was reduced from 0.099 percent sulphur to 0.022, and another from 0.088 to 0.018 percent. This possibility of so low sulphur in gray-iron castings may possibly open up a new field for metallurgical investigation.

Carbon regulation, total as well as combined and graphitic, is possible to a most useful extent in the electric furnace. By placing steel scrap in the furnace before adding the molten iron from the cupola, total carbon can with accuracy be reduced to any desired extent. By this simple, unfailing means, gray iron of the so-called semi-steel quality may be obtained with great uniformity of composition, structure, and strength. Uniformness among different heats, homogeneity in the individual heat, close carbon regulation, and unlimited temperature, are a quartet of benefits

of the electric furnace that make possible a realization of those exceptional qualities so often claimed for and so rarely reached by those cupola-compounded mixtures that are pleased to go under the name of semi-steel. Correctly balancing the combined and uncombined carbons also is a matter of no great intricacy in the electric furnace by the vicarious means of the silicon content. Indeed, the capricious vagaries of the cupola with reference to carbon control, are replaced in the electric furnace by substantial certainties.

Another advantage accruing from the electric furnace is the possibility of making furnace additions of manganese and other elements for particular purposes, without having to expect more or less appreciable losses; the manganese losses are *nil*. Other benefits there are, but they are of a well-known order, so that their repetition here would be commonplace. The one big advantage from the standpoint of composition is the absolute control of mixture, making duplication of results more a matter of correct calculation and less the effect of happy accident.

A concrete example of results actually obtained in everyday running practice may be of interest. A rather ordinary mixture of pig iron and foundry scrap was melted as usual in the cupola, transferred to a basic electric furnace and there superheated and refined under a lime slag. The untreated iron was of a composition regularly giving standard "arbitration" test bars that broke under an average transverse load of 2,950 pounds (1,350 kg.) with a deflection of 0.10 inch (0.25 cm.). After treatment in the electric furnace, the iron gave the same kind of bars breaking at slightly over 4,400 pounds (2,000 kg.) and at a deflection of 0.115 inch (0.29 cm.). The specific gravity was increased from 7.10 to 7.25. About twenty-five minutes was the time of the electric furnace treatment, and the current consumption was 104 kilowatt-hours for 2,000 pounds (910 kg.) of iron.

At the beginning of the paper the author indicated several concrete cases which seemed to justify this proposed dual process, and now before closing he wants to define broadly some classes of iron castings for which it may be feasible. In the first place, there are those castings for which there is exacted unusual tenacity, solidity, and other physical properties. Secondly, there are those castings that are difficult to "run" on account of having

thin sections and relatively large sizes. Thirdly, there are those castings of high quality whose extreme foundry cost is but a very small part of the total cost of the finished article. For the moment, leaving the cupola, attention may well be called to possible advantages in using the electric furnace in connection with the "direct metal" from the blast furnace, the electric furnace filling the triple rôle of mixer, superheater, and refiner. Such process would make possible high-class gray-iron castings almost direct from the blast furnace at a cost very little higher than that of ordinary cupola iron.

To conclude, the author wants to emphasize the fact that the cupola and electric furnace duplex process for gray iron is not recommended by him as a commercial practicability for ordinary iron castings under ordinary circumstances, but rather as a convenient and extremely efficacious substitute for the cupola process for extraordinary castings or for extraordinary circumstances.

DISCUSSION.

Richard Moldenke[1]: I want to compliment the author on his paper, because I think he has cleared up a few points that have been bothering foundrymen a good deal. In an experience of about 30 years that I have had in the foundry, my memory goes back to the sulphur situation as it was in cast iron scrap in the early days. Scrap has to be used over and over again, and unless diluted with plenty of good pig iron will result in poor castings. Hence cast iron scrap is the important thing we have got to watch. About thirty years ago we had our sulphur in the scrap at 0.05. Today I think you will find that the scrap runs at least 0.12 in sulphur. It has been creeping up by the constant remelting of the foundry scrap until it is even now quite a serious problem. If the situation keeps on the way it has been going during the war, when the coke furnished was so high in sulphur that castings with 0.30 sulphur content were quite common, you can see what we are up against, particularly as these high sulphur

[1] Consulting Metallurgist, Watchung, N. J.

castings will all come back to the foundry as scrap again. Anything that will show us a chance to get the sulphur down in cast iron is to be welcomed; in fact we are going to face a situation where we will have to get the sulphur down in cast iron, perhaps in our lifetime yet. The paper indicates that if we transfer and hold the molten metal, in the electric furnace with a basic hearth, we can get the sulphur down to a third or a quarter of what it was. That is an opportunity we welcome. The author said a few words about sulphur's advantages and disadvantages. One advantage sulphur has in the foundry is in making a metal that shows good wearing qualities, as in cylinders. Again, high sulphur in castings results in a granular sulphide of iron surrounding the crystals of the purer iron. Consequently in sliding friction, under high pressure we have a better braking surface to go against than when the sulphur is low. The brake shoe industry has found this to be the case: With a very high sulphur brake shoe a beautiful braking action is exerted on the car wheels, because the shoe crumbles to powder and therefore brakes tremendously, even if it gets red hot. The very fact, however, that it gets red hot makes the high sulphur shoe dangerous and it should not be used. The other point in high sulphur that you may be familiar with is in cutting threads. Whenever you have a very pure metal, you can hardly cut a thread successfully because it tears badly. With the high sulphur, the threads cut clean and nice. So it is only in machining or wearing problems that high sulphur has a little advantage. We put up with it because we cannot help ourselves, but we would rather not have the sulphur.

The question of using the electric furnace for grey iron work has come to my attention professionally in four cases during the last six months. My clients were anxious to use the electric furnace in every case if it was at all possible, but they found the cost was a serious question. My investigations indicated that electrically molten iron should be attainable for about half a cent a pound over the cost of cupola molten iron. Suppose you are producing very small castings which actually cost you in the foundry some eighteen cents a pound to make; what difference does it make if you add half a cent a pound to it? The electric furnace ought to go into such a foundry, and go in quickly. It takes some courage and considerable cash to do the thing, but

where a casting costs over say ten cents a pound to manufacture, the extra half cent added for the electric melting process does not cut any figure at all, because it is gotten back from the customer, anyhow, for the additional quality obtained. Referring back to the first meeting at Niagara Falls at which the question of electric steel came up, I remember stating that it required almost nine times as much heat to bring cast iron up to the melting point as to actually melt it, not considering the subsequent superheating: in other words, if you can get steel or iron white hot just before it melts by some cheap method of using coal, then the cost of electric power for actual melting thereafter is only a small item. I proposed at that time if the steel men would take ordinary Bessemer steel and get it white hot in soaking pits as fired ordinarily and then use the electric furnace for melting, the electric furnace would be further along than it is today. This is what we want to get at today. The foundryman can get this molten metal from the cupola very cheaply, but he has got a lot of sulphur into the iron this way and cannot get it out again. The basic lined electric furnace offers this opportunity and the combination of cupola and electric furnace will result in comparatively cheap melted cast iron of low sulphur content.

Now there is another little question—about the carbon. I see Mr. Elliott speaks about "semi-steel," so called. I have been fighting this designation all my life. I do not think I am succeeding very well in fighting it, but I feel we are all seekers after truth and speaking in general, whoever calls a material "semi-steel" when he should know it is cast iron and has none of the characteristics of steel at all, is not following my idea of the scientist who is looking for the truth wherever he finds it. Commercially we run across the following fact: one foundryman uses five percent steel in his mixture and another foundryman uses fifty percent; and both call it "semi-steel." Why? Simply to get one or two cents a pound more out of his customer. Therefore I feel the name is a fraud. Now Mr. Elliott tells us that in the electric furnace you can regulate the question of carbon in the casting absolutely. This is valuable, because such a regulation is impossible in the cupola. The electric furnace, basic lined, may make it possible to get what the foundryman seems to be looking

for, namely: a separate class for a very high-grade cast iron, based upon a definite carbon content.

There are some little points upon which I do not quite agree with Mr. Elliott; for instance, the question of superheating the iron in the cupola. If the cupola melting was carried out correctly, the molten metal is obtained much hotter than we can get the same metal in the air furnace or even the open hearth. I have made many heats by all three melting methods, and have always found that the cupola iron, when run right, is the hottest metal you can make. If we attempt to melt cast iron at as high a temperature in the open hearth furnace, we will get an oxidized product that yields defective castings. In the cupola process, you want to melt cast iron at the highest possible temperature the cupola will give, and then you get the best iron. In the furnace, however, you have got to work the other way. It is necessary to melt and gradually heat up to the desired point of superheating. Every ten minutes additional means damage by oxidation. Hence in the open hearth furnaces you cannot go to the high temperatures you can get in the cupola. Here, evidently, the electric furnace will help, as it gives the highest degree of preheated iron with the least oxidation. Finally, in the foundry we are not looking for any refining action; we want a plain melting action. You do not want to refine, because the moment you do this, you introduce oxygen, and when you introduce oxygen, you cannot get it out again and the result is gas holes and excessive interior shrinkage, etc., in your castings, which you do not want; the molten metal loses its "life," you cannot wait before pouring it. The freezing point of the metal goes up, the casting sets irregularly, some metal remaining liquid within, which shrinks because you cannot feed more freshly molten iron. Grey cast iron, in going from the molten to the solid state, loses about 5.8 percent in volume. When you have poured molten iron into the mold you have got to keep feeding more iron in or you are bound to have unsound spots in your castings. I do not believe there is such a thing as a perfectly sound grey iron casting in existence. White cast iron is even worse, as the reduction in volume is more than twice that of grey iron. In the electric furnace it is quite possible to do some refining, because a deoxidizer can be added.

G. K. Elliott: It has indeed been both a pleasure and a privilege to me to listen to Dr. Moldenke's remarks concerning the paper I have just read, even though we do not seem to be entirely agreed upon several of the points discussed. One of these is the subject of hot iron and there I believe the Doctor has somewhat misunderstood my meaning. That a well-managed cupola will deliver relatively hot iron is absolutely true, but it is also equally certain that there is a limit to the superheating capacity of even the most effectively operated cupola, a limit that for some irons falls short of ideal conditions. In my paper I refer exclusively to degrees of superheat that are physically impossible in any cupola type of furnace. Fifteen years' experience in iron foundries making a specialty of valves and other forms of pressure containing castings, all of which had to be solid and entirely leakless at water and steam pressures of hundreds of pounds, has convinced me that hotter iron than the best of cupolas is capable of giving, produces castings that are uniformly more solid and that pass the hydraulic pressure tests with a minimum percentage of leakers. The exact relation between various degrees of hot iron and closeness of grain, is a matter readily determined in any foundry making pressure containing castings, valves for example, because all such castings are invariably tested under fluid pressure before leaving the shop, and the foundryman has always before him definite, practical data concerning the solidity of all his castings. It is almost as though this foundryman were casting nothing but test bars, all of which are being tested and reported back to him.

As to refining, I agree completely with Dr. Moldenke in viewing with alarm the introduction of oxygen into cast iron even though there be metallurgists who advocate "oxygenated" iron. However, refining in a basic bottom electric arc furnace, as I have proposed, is emphatically not an oxidizing operation; on the other hand it is decidedly deoxidizing. So excellent have been the results that I have seen obtained from grey iron refined under the reducing conditions of the electric furnace, that I cannot be other than a firm believer in the superior qualities of deoxidized iron as well as a confirmed skeptic concerning the claimed advantages of oxidized or "oxygenated" iron.

RICHARD MOLDENKE: Perhaps Mr. Elliott and I are in agreement, after all. I did not refer to the electric furnace in discussing superheating, but more particularly about the air furnace and the open hearth furnace; if the electric furnace superheats the metal more than the cupola, so much the better. I did object to superheating molten cast iron in the air furnace and the open hearth furnace on the ground of danger from oxidation. The more we deoxidize, the cleaner and purer we get the iron in the casting. It remains fluid longer and has a better chance to feed in the mold. The trick in the molding is to try to get the iron into the mold through the gates directly to an end point and then, if possible, let the metal set backwards from this end point to the gate before the gate itself has set and thus closed off the supply of molten metal. Then you get a reasonably dense casting, and you can not get it thus otherwise.

J. W. RICHARDS[2]: If I understood Dr. Moldenke rightly, he says that all there is to do in the foundry is to melt, superheat and cast, but not to refine. Do you here use the word "refining" as equivalent to oxidation?

RICHARD MOLDENKE: Yes. I use the term "refining" in the iron master's sense, where it refers to the oxidation of silicon, manganese and carbon.

J. W. RICHARDS: But this overlooks the fact that the refining done in the electric furnace is done by reduction, not by oxidation; the removal of sulphur is done in the presence of lime in a strongly reducing atmosphere; there is no danger of oxidizing the iron while you are reducing out the sulphur, so it does not bring with it that train of evils which you fear.

[2] Prof. of Metallurgy, Lehigh University.

Presented as part of a Symposium on "Released Electrochemical Information" at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

DIRECT PRODUCTION OF "WHITE ARSENIC."

By Charles A. Doremus.[1]

(Preliminary Note.)

During 1906, 1907 and 1908 carload shipments of hand-picked arseno-pyrites, "Mispickel" were made from a mine at Carmel, Putnam County, New York, to Swansea, Wales. A certificate of analysis by Ledoux & Co., of a shipment of 259,337 lb.. (117,750 kg.) showed:

	Percent
Arsenic	21.68 percent
Sulphur	14.44 percent
Iron	22.19 percent
Insoluble	41.08 percent
Gold and silver	trace

Recalculated this ore would appear as a mixture of 47.10 percent of AsSFe, and 9.67 percent of FeS_2.

Other carload shipments showed As from 23 to 31 percent. Owing to large deposits of similar ore in Saxony, and Bohemia, this exportation ceased.

The increasing demand for "white arsenic" in this country, primarily for insecticides, and later for poison gases in war offense work, induced Commercial Chemicals, Inc., of New York, to be organized, lease the mine, with purchase option, and develop a plant at Mamaroneck, New York, for roasting the ore.

The site at Mamaroneck was chosen because of the existence of a rotary kiln which could be quickly put into commission, the ease with which the ore could be transported to this point by rail, or motor truck, and the good labor and supply conditions.

The apparatus consists of an internally heated rotary kiln 20 feet (6 m.) long, and 4 feet (1.2 m.) in diameter. The exhaust gases pass to a Sturtevant cyclone dust collector, and then on

[1] Consulting Chemist, New York City.

to a Cottrell precipitator. This was especially constructed to care for 4,000 cubic feet (112 cu. m.) of exhaust per minute, with a 6 linear foot (1.8 m.) velocity in the tubes per second, at 200° F. (94° C.). The precipitator is in two sections, each having 16 8-in. (20 cm.) pipes 15 feet (4.5 m.) long, in the center of each a dependent chain. When the collected "white arsenic" is shaken from the pipes and chains it falls into a trough at the bottom of each section, and is carried by a screw conveyor to a common bin, and from here directly to a barreling machine. This avoids raising dust, and makes a highly sanitary plant.

The kiln was heated by a series of flames obtained by mixing gas and air in a 4-in. (10 cm.) steel pipe provided with some 80 ¼-in. (6.3 mm.) nozzles. The flames were directed, owing to the air pressure, against the surface of the ore, and at right angles to the length of the retort.

This procedure avoided the making of dust, and the exhaust going to the dust collector was clean, and transparent, for its temperature was above the vaporizing point of the "white arsenic."

The ore was charged to a bin, and entered the retort at the end next the discharge to the Cottrell precipitator through a screw conveyor.

The cinder was automatically discharged at the opposite end, and the mechanical operation of the plant was so simple that only a foreman and a helper were needed for each of the three shifts. Other retorts might have been run with no increase of this force.

The entire plant was run on a closed draft system, controlled by a Sturtevant fan placed beyond the Cottrell precipitator, and drawing the air through the kiln, dust collector, Cottrell precipitator, and discharging the exhaust into a brick chimney whose top was above the building.

The Cottrell precipitator was provided with suitable electrical apparatus for converting the current from the local power system into the D. C. high-tension current required for precipitation. Most of the time when running this current was of 65,000 volts.

Under proper attention the exhaust above the chimney was "optically clear." Even in the brightest sunlight no dust was visible. This condition existed under the running given, which was not the maximum capacity, but it is reasonable to believe that it would be the same when maximum is reached.

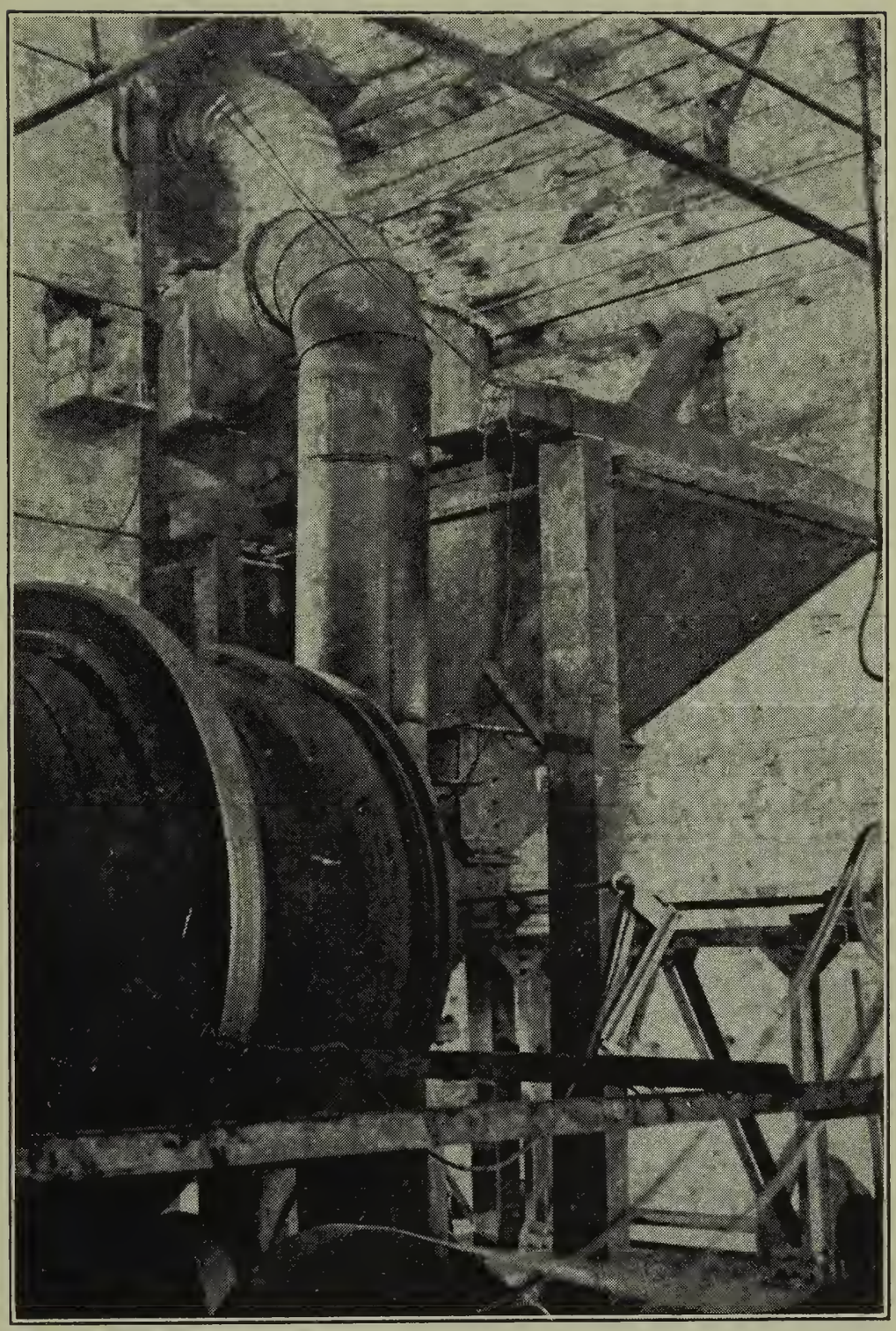


Fig. 1. Feed end of Rotary Kiln.

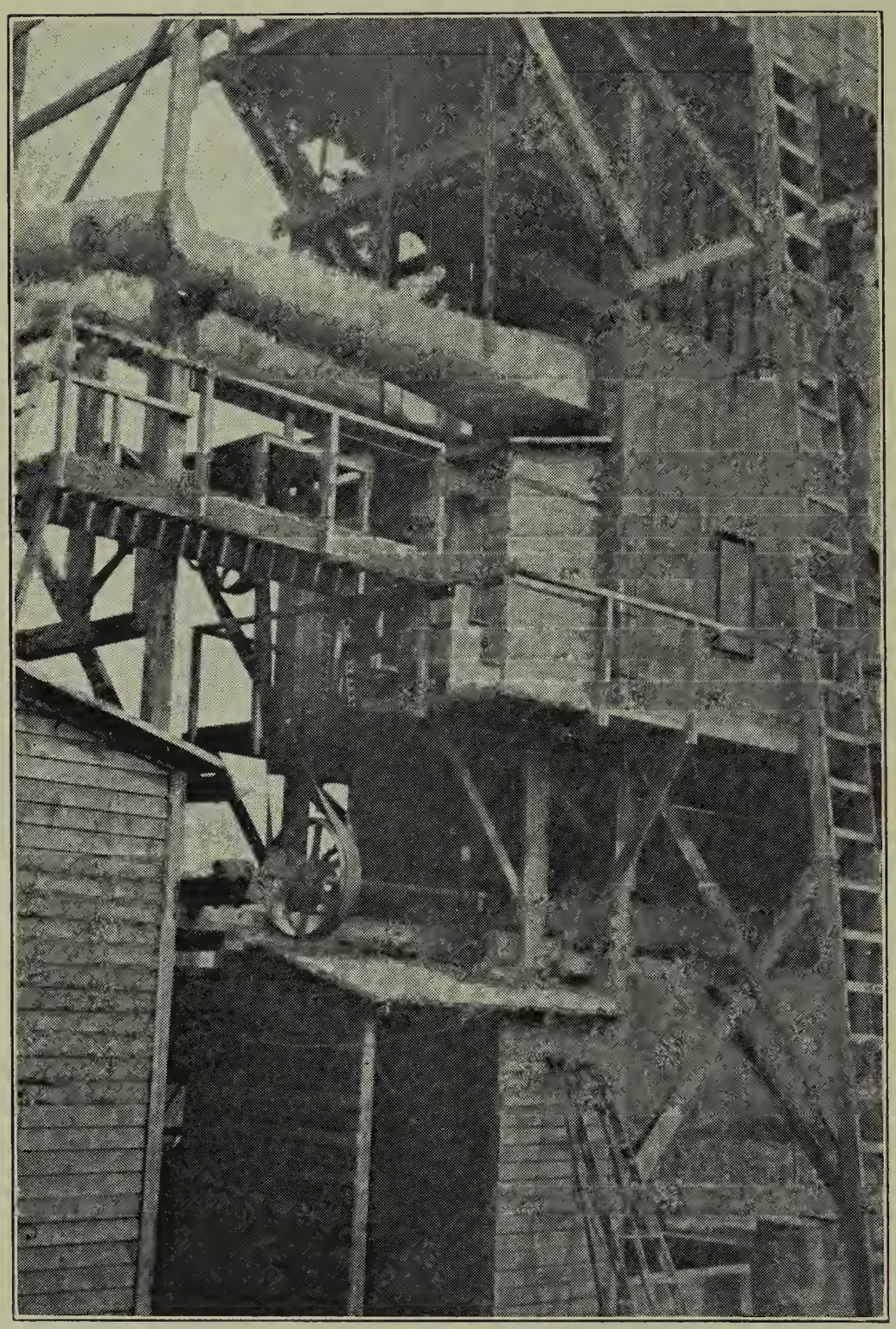


Fig. 2. Pipes leading the exhaust to the Cottrell precipitator.

It was found after a short time running on hand-picked ore that a concentrate would be preferable. A "Buchart" table was set up at the mine, and concentrate of the following composition obtained, as per analysis of Ledoux & Co.:

Arsenic	29.52 percent
Sulphur	18.65 percent
Iron	29.15 percent

This would indicate a mixture of about 64 percent AsSFe, arseno-pyrite, with 11 percent FeS_2, pyrite. The quartz gangue would be about 25 percent.

When the elementary arsenic is oxidized, it combines with about one-third its weight of oxygen. A 30 percent concentrate would therefore yield 40 percent As_2O_3, and a short ton of 2,000 lb. (908 kg.) give 800 lb. (363 kg.) "white arsenic." The Government fixed the price of this at 9 cents per lb. (20 cents per kg.).

The product collected directly from the runs, without refining, analyzed 98 percent + of As_2O_3. There was 0.4 percent dust, and a slight amount of SO_3, which, of course, became H_2SO_4 through absorption of moisture. The product was barreled and sold at maximum price.

Negotiations were in progress at the time of the signing of the armistice, looking to a greater production to supply the Government with this then much-needed chemical. Since then, to meet peace conditions, the mine is being developed, a larger concentrating table has been installed, and the kiln is being tried out to roast the ore without aid of fuel, relying on the content of arsenic, sulphur and iron for the production of the necessary heat.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

URANIUM: ITS EXTRACTION FROM PITCHBLENDE.[1]

By Gustave Gin.[2]

Abstract.

A discussion of the occurrence and composition of pitchblende (Uraninite), the methods of treatment by which uranium oxide is separated from it when it contains vanadium, the simplified method for use when vanadium is absent, and a proposed method of Mr. Gin for purification, involving fusion and electric furnace treatment, to convert most of its constituents into carbides, followed by separations based on the differing properties of the carbides. [J. W. R.]

ORES.

Pitchblende is the most abundant of the minerals containing uranium. It is practically the anhydrous oxide, U_3O_8, mixed with oxides of molybdenum, tungsten and vanadium; it contains also silica, lime, magnesia, baryta, sulphide of lead, and the oxides of iron, manganese, nickel and copper. Its density varies between 7 and 9; hardness 5 to 6; it crystallizes in cubic forms but the crystals are difficult to distinguish. It is most frequently found in mammillary masses of a greenish or blackish pitchy luster, which gives it its name. It is very difficult to fuse.

Pitchblende contains radio-active materials, the most important of which is radium, discovered by M. and Mme. Curie. There is also in it another stronger radio-active substance which has

[1] This short communication was sent by Mr. G. Gin some years ago, with the statement that it would be followed by a longer dissertation on this metal, corresponding to those which he had furnished to the Society on Molybdenum (Transactions (1907), **12**, 411) Tungsten (Trans. (1908), **13**, 481), and Vanadium (Trans. (1909), **16**, 393). Nothing further on Uranium was received, but the present increasing interest in this metal is thought to justify the publication of this short note. (Translated from the French by Jos. W. Richards and W. S. Landis.)

[2] Consulting Electrometallurgical Engineer. Now deceased.

been discovered by M. Debierne, and to which he gave the name of actinium. Mme. Curie has also extracted from the residues of pitchblende another radio-active body: polonium.

The principal occurrence of pitchblende is in Joachimsthal in Bohemia. It is also found at Przibram (Bohemia), Johanngeorgenstadt and Schneeberg (Saxony), Rezabanya (Hungary), Redruth (Cornwall), Adrianople (Turkey), Canada, and various places in Colorado.

The hydrated varieties of pitchblende comprise: Gummite, found in yellow or reddish brown amorphous masses looking like gum and containing 4 to 5 percent silica, 5 to 7 percent lime and 14 to 16 percent water. Eliasite is an impure hydroxide of uranium, amorphous, of greenish color.

Uranium is also found as oxide in Cleveite, in which was found for the first time the element helium. It is found at Carlshuus in Norway; it is a uraniferous spinel containing lead, cerium, yttrium, etc.

A new mineral containing uranium was studied in 1899 by Friedel; it is a double vanadate of uranium and potassium and is called Carnotite, $K_2(UO_2)(VO_4)_2.3H_2O$. This rare mineral is of great interest, for there have been discovered some large deposits of it, which may become important in the production of vanadium and uranium. It was discovered in 1895 by Poulot, in Montrose County, Colorado, filling cavities in sand-stone. It is accompanied by carbonates of copper. It is in yellow masses, pulverulent, slightly crystalline, and mixed with quartzose sand. It has also been found in Chile and Brazil.

TREATMENT OF PITCHBLENDE.

The treatment of pitchblende is necessarily complex and it is impossible to give complete details, for the methods of separation and elimination of impurities must be adapted to the composition of the material under treatment. At Joachimsthal they have used the methods of Arfvedson, Peligot, Wöhler, Schafarick, Ebelman, Kessler, and Anthon.

The process which has been last used to prepare simultaneously the green oxide of uranium and vanadic acid is as follows: Lime is added to the ore and it is roasted in a reverberatory furnace, with an oxidizing flame. The uranium passes to the state of

neutral calcium uranate; vanadium, tungsten and molybdenum are likewise transformed into calcium salts.

$$2U_3O_8 + 6CaO + O^2 = 6CaUO_4.$$

The calcined mass is cooled, pulverized, and treated with dilute sulphuric acid. Uranium, tungsten, vanadium and molybdenum pass into the solution, which is filtered through asbestos cloth. Ferric chloride is added to the solution, which is then neutralized with carbonate of sodium forming insoluble vanadate of ferric oxide, while the uranium is not precipitated. There are thus obtained a precipitate containing vanadium and a solution containing the uranium, which must be treated separately.

The solution is evaporated to dryness and taken up with a mixture of water and alcohol, which dissolves only the uranium and leaves insoluble the phosphates and sulphates of the alkaline earths. From the new solution the uranium is precipitated as ammonium uranate by the addition of ammonia. The precipitate is voluminous and yellow; it flocculates to a pulverulent mass on standing hot. It is filtered and washed with water containing ammonium chloride. The ammonium uranate is utilized in industrial processes under the name of lemon-yellow uranium (jaune citron d'urane). It is used for the same purposes as sodium uranate.

The precipitate of ferric vanadate is fused with a mixture of carbonate and nitrate of sodium. It is then taken up with boiling water and precipitated by gallic acid. The black precipitate is washed, and gives on calcination sodium vanadate contaminated with arsenite, arsenate, molybdate, tungsten and silicate of soda. It is redissolved, reduced by sulphurous acid, and heated to eliminate excess of sulphur dioxide; a current of hydrogen sulphide is then passed through and precipitates the arsenic and molybdenum.

To remove the rest of the tungsten and silica the solution is evaporated to dryness, dissolved in sulphuric acid, neutralized by sodium carbonate oxidized by chlorine, and the vanadium finally precipitated as ammonium vanadate in a cold solution in the presence of an excess of ammonium chloride. The precipi-

tates containing the tungsten and molybdenum are of little importance. They are allowed to accumulate and treated separately when there is enough on hand.

As is seen, the process is extremely complicated and necessarily tedious, and there are such a large number of precipitations, filtrations, etc., that the process cannot be left to unskilled workmen without the constant supervision of a skilled chemist.

I am not certain that the method indicated has not been changed at Joachimsthal since the discovery of radium has given a considerable value to the residues.

ANOTHER METHOD OF TREATMENT AT JOACHIMSTHAL.

There has also been used at Joachimsthal for minerals not containing vanadium in sensible amount a more direct process.

The pitchblende containing about 40 percent U_3O_8 is crushed in a jaw crusher and then ground fine in a ball mill. The powder is charged in amounts of 150 to 200 kilograms into a reverberatory furnace with oxidizing flame, to remove sulphur and arsenic and peroxidize the uranium to a mixture of olive-green oxide U_3O_8 and sesquioxide UO_3.

The roasted material is cooled and mixed with 20 percent of a mixture of equal parts of nitre and carbonate of soda. This is then heated to redness and kept there two hours with strong stirring. The reactions are as follows: During the first part of the heating sulphur and arsenic are oxidized and the uranium is completely oxidized to sesquioxide; the treatment with nitre of soda forms sodium uranate as well as vanadate, molybdate, tungstate, arsenate, silicate and phosphate, etc., of sodium.

The material is cooled, finely pulverized, and digested in filter vats similar to those used in the hyposulphite treatment of silver ores. Here it is washed methodically with hot water until the filtered solution shows no precipitation with barium chloride. The vanadate, molybdate, tungstate, arsenate, silicate and phosphate of sodium are thus removed, while sodium uranate remains in the residue with certain other metallic oxides.

The residue is treated with mono-hydrated sulphuric acid mixed with 5 percent nitric acid, in order to precipitate the last traces of the lower oxides of uranium. After two or three hours' attack the liquid is diluted with water and the silica is allowed

to deposit, together with a part of the oxide of iron and almost all the calcium sulphate. These are separated by decantation. The decanted liquid, containing the uranium as uranyl sulphate, is treated with sodium carbonate in excess. All the metals, such as iron, cobalt, nickel, copper, etc., which have followed the uranium so far, are precipitated, while the uranium is dissolved in the excess of the reagent as the double carbonate of sodium and uranyl, which is soluble in water. The strongly yellow solution remaining is used for the manufacture of the clear yellow uranate or the orange-yellow uranate.

The clear yellow uranate is made by boiling the solution in an enamelled cast-iron pan and adding from time to time sulphuric acid, 10° B., to drive off the carbonic acid. When the neutralization is exact all the sodium uranate precipitates and the liquid will not give a further reaction for uranium with potassium ferrocyanide.

The orange-yellow uranate is obtained by precipitating the double carbonate in a boiling solution with caustic soda. The green oxide is obtained from the sodium uranate by boiling with a strong solution of ammonium chloride, forming sodium and sodium uranate. The latter on being calcined gives the green oxide.

GIN'S PROCESS FOR THE TREATMENT OF PITCHBLENDE.

The powdered mineral is roasted in a reverberatory furnace until sulphur is all removed. Almost all the arsenic is likewise removed and the uranium changed into sesquioxide.

The roasted material is mixed with carbon and put into an electric furnace with two hearths. The quantity of carbon introduced should be insufficient to reduce all the metallic oxides. The lead, copper, iron, nickel, cobalt, and perhaps a little of the tungsten, will be liberated as metals, and there is formed a slag containing silica and the lower oxides of uranium, vanadium, tungsten and molybdenum. The base metals are run off and carbon is then added to the slag in sufficient quantity to reduce all the oxides present and transform them into carbides.

The further treatment in the electric furnace gives a mass of carbides of uranium, tungsten, vanadium, and molybdenum, which if cooled quickly on removal from the furnace can be easily pul-

verized. The powdered carbides are treated with water at 60° C. Uranium carbide decomposes water, giving hydrated uranium oxide, green at first, but rapidly becoming blackish on contact with air. This precipitate is separated by flotation and dried and calcined. It then serves for the electric preparation of the pure metal or of its alloys or for the manufacture of uranium compounds such as the nitrate of alkaline salts.

The residue from the treatment with water contains all the unattackable carbides and silicides. It is dried and treated with potassium nitrate to fusion resulting in a mixture of the alkaline vanadates, tungstates, molybdates and silicates. This fusion is dissolved in hot water and hydrochloric acid added. Tungstic and silicic acids are precipitated while molybdenum and vanadium remain in solution with any phosphoric acid. After filtration and washing of the precipitate tungstic acid is separated from the silica by fusion with bisulphate of potassium. The product of this fusion is dissolved in water, which removes potassium sulphate and leaves a white residue of tungstic acid and silico-tungstate of potassium. The residue is treated with cold water to which is added ammonium carbonate, dissolving the tungstic acid and leaving the silico-tungstate undissolved.

In the solution coming from the first operation and containing molybdenum and vanadium it is extremely difficult to remove phosphoric acid, and it is preferable in practice to precipitate it as ammonium phospho-molybdate. The solution is concentrated and ammonia is added until a yellow precipitate is no longer formed. Filter, and in the filtrate the vanadium present may be precipitated as ammonium vanadate by adding ammonium carbonate and crystals of ammonium chloride. The filtered liquid contains only ammonium molybdate, that is if all the molybdenum has not previously passed into the phospho-molybdate. The ammonium molybdate is obtained by concentrating to dryness and calcined in the air to molybdic acid.

Finally molybdic acid may be separated from the phospho-molybdate by dissolving it in a concentrated ammoniacal solution and adding chloride of magnesium to precipitate the phosphoric acid as ammonium-magnesium phosphate.

DISCUSSION.

S. C. LIND[1] (*Communicated*): The postponed publication of Mr. Gin's paper at the present time lends to it an added interest in affording an opportunity of looking backward at the rapid changes that have taken place during the past ten years in the metallurgy of some of the rare metals. Perhaps no better illustration could be found of a completely changed situation than by reference to uranium.

Mr. Gin's opening statement that "Pitchblende is the most abundant of the minerals containing uranium" is no longer valid. Carnotite to which he referred as "the rare mineral of great interest which may become important in the production of uranium or vanadium" has in the meantime so far outstripped pitchblende as a source of uranium (and radium) that the 1918 production alone from carnotite is as great as the total from pitchblende since the time of Mr. Gin's prediction. This is, of course, due to the development of the well-known carnotite fields of Colorado and Utah.

A still more important change has taken place. Uranium, instead of being the metal of primary importance in uranium minerals, has now become the by-product while radium has become the primary product whose recovery is sought in spite of its infinitesimal concentration of only one part to three million of uranium. This fact has naturally changed the metallurgical situation; and any process for uranium which does not lend itself to the recovery of radium must be discarded. This can not be overemphasized. Uranium is not only at present a by-product, but one that must struggle hard to find a commercial outlet; only a small portion of the uranium being produced in the radium industry is finding a market at the present time.

With respect to actual metallurgical processes now in use for uranium minerals, while a large number have been published or patented in the past ten years, the method of the Austrian Radium Commission, by Haitinger and Ulrich[2] can be regarded as standard for pitchblende. For carnotite, the processes in use divide themselves into two general classes: First, those using an initial

[1] U. S. Bureau of Mines, Golden, Colo.

[2] Haitinger and Ulrich, K. K. Akad. Wiss., **117**, Oct. 9 (1908).

acid leach in the absence of sulphuric acid, thus separating the radium at once from the gangue material, as for example, the nitric acid process of the U. S. Bureau of Mines.[3] And, second, those using an initial sulphuric acid[4] or acid sulphate[5] treatment which puts the radium into the tails from which it must be slimed off by some mechanical process. Perhaps a third general class of methods should be mentioned, treating the ore initially with soda ash, either fused or in solution, leaving the radium with the gangue material as carbonate subject to acid leaching, while dissolving directly the uranium and vanadium. Many variations of these principles are possible in practice. In nearly all present processes for uranium it is first recovered by precipitation as the "yellow oxide" ($Na_2U_2O_7$) which has a few very limited uses. The conversion of this product into the black oxide (UO_2) may be very simply and economically accomplished by the process of Dr. Parsons of the U. S. Bureau of Mines.[6] The black oxide is the product usually preferred metallurgically in the preparation of ferrouranium or uranium steel. The non-ferrous alloys of uranium have been but little investigated.

[3] U. S. Bureau of Mines Bulletin 104, Parsons, Moore, Lind and Schaefer.
[4] McCoy: U. S. Patent, 1,098,282.
[5] H. Schlundt: U. S. Patent, 1,240,607.
[6] C. L. Parsons, J. Ind. and Eng. Chem. (1917), 9, 466-7.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

NOTES ON ELECTROSTATIC PRECIPITATION.[1]

By H. D. Braley.[2]

Abstract.

A technical and industrial review of the general question of electrostatic precipitation of dust and fume, including the removal of emulsified water from oil. The various applications discussed are: recovery of metallic dust, of antimony, sulphur and arsenic, of potash, of sulphuric acid mist. Figures of costs of various plants, expense of operation, values recovered, and profits secured, are given in detail. Separate discussions are given regarding the construction and operation of various types of treaters, temperature control as affecting operation, gas velocity during treatment, direction of gas flow, capacity of electrical supply, operating voltage, electrical apparatus, transformers, generators, synchronous motors, rectifiers, switchboard and accessories. [J. W. R.]

The application of electrostatic precipitation principles to commercial processes has grown so rapidly within the last few years, and the field has broadened to such an extent, that it is not unexpected that many of us lose sight of its many and varied applications and the importance of the position it has assumed in its relation to our industrial activities. There has been a number of excellent papers and reports written in the past on certain special phases of the subject, most of which have, however, dealt only with a particular application of the method.

It is the object of this paper to present, as far as possible, a review of the applications of the electrical precipitation process,

[1] Manuscript received Sept. 13, 1918.

[2] General Engineer, Westinghouse Electric & Mfg. Co., East Pittsburgh, Pa.

with a brief description of the particular features developed for each, its importance as an industrial factor both from the standpoint of a paying financial investment and as a conserver of our natural resources, and finally to suggest the possibilities of new applications, where it appears that profitable development may ensue.

Before proceeding further we shall review briefly the fundamental principles upon which the electrical precipitation process operates, so as better to establish their connection with the practical applications.

When finely divided particles are introduced into an electrostatic field they become charged and are acted upon by forces which propel them in a direction determined by the field. This action is tersely explained by the two well-known fundamental laws of electrostatics, the first of which is that bodies having like charges repel each other, and those having unlike charges attract each other, and second, that the force exerted between two electrically charged bodies is proportional to the product of the strength of their charges and inversely proportional to the product of the squares of their distance and the dielectric constant of the medium separating them.

In the treatment of fume carrying suspended matter, the first law is illustrated by the imparting of electrical charges to the suspended particles, with the result that the particles are propelled to a surface suitable for their retention and eventual collection, while the second law is translated into terms of voltage maintained on the electrodes, distance between electrodes and the dielectric qualities of the gas being treated, the latter varying with the composition, temperature, etc.

Numerous refinements of treater construction and operating practice have contributed to the success of the process; among these may be mentioned: design of electrodes with regard to controlling the point of deposition of the particles, gas velocity by means of proper proportioning of precipitating tubes, location and design of dampers and draft control, selective treatment by temperature control, dry and wet treatment of gases, and special characteristics of electrical generating and transforming apparatus, etc., etc.

Applications and Recovery of Fume.

The chief present-day application of the electrostatic precipitation process for the recovery of values may be classed:

(a) Recovery of metallic dust from gas fumes given off from ore smelters, roasters and blast furnaces, the most important of which are lead and copper recoveries.

(b) Recovery of antimony, sulphur and arsenic from copper and lead-bearing ores.

(c) Recovery of potash from cement dust in cement mills treating potash-bearing limestone.

One of the first applications of the electrostatic precipitation process was that employed for the precipitation of sulphuric acid fumes given off from lead smelters, and it is interesting to note that the primary reason for the development was not for the recovery of the acid but rather for the abatement of a nuisance —the recovered material being considered as of no particular value at the time and with some considerable justification, as there was then no such demand for this recovery as that which exists today. Closely following this first installation, others have followed, until at the present time the plants for the recovery of metals, especially copper and lead, have reached relatively immense proportions.

Financial Considerations.

A typical installation of this process is that of the International Smelting Co., although numerous others could be given, notably those of the Tacoma Smelting Co. treating lead-bearing ores, the Anaconda Copper Co. treating copper ores, the Consolidated Smelting Co. treating lead-bearing ores, and the St. Joseph Lead Co. treating lead ores.

Authentic figures for the plant of the International Smelting Co. are available and have been so concisely and clearly presented by Mr. Walter A. Schmidt in his paper on "Dust and Fume Collection by the Cottrell Process of Electrical Precipitation," that they are given here, and as before stated, they are typical of what has been done and of what may perhaps in general be expected from a large precipitation plant.

There are two electrical installations at this smelter, one of which treats the gases coming from the roasters used for drying flotation concentrates and the other for treatment of fume from copper converters. The figures given, however, cover the complete installation.

First Cost:

Structural steel and tubular electrodes	$83,500.00
Electrical equipment	20,900.00
High tension wiring and accessories	9,500.00
Total installation cost	$113,900.00

Recovery:

Drier plant 5 tons dust (25% Cu)	(1110 kg.)	2500 lb. copper per day
Converter plant 2.3 tons dust (35% Cu)	(730 kg.)	1610 lb. copper per day
Total copper recovered	(1840 kg.)	4110 lb. copper per day
Value at 12 cents per lb.		$493.20 per day
Per annum		$180,018.00

Operating Expenses:

Operating costs, labor, power, and expenses ($40 per day)	$14,600.00
Gross surplus per annum	$165,418.00

The figure $165,418.00 covers amortization, interest and profit. Recoveries are based on normal low price of copper at 12 cents per pound for form in which it is recovered in precipitators. Profit based on copper values only.

The operating crew consisted per shift of one operator and one helper who attended electrical control room, shaking down of electrodes, electrode cleaning and other incidental operations. The power consumption for the total plant amounted to less than 40 KW. It should be noted that no dust collection flues were installed between furnaces and treater, the former discharging directly into the treaters. The treater installation costs represent entire cost except the converter stack.

A considerable amount of additional material could probably have been obtained by putting in settling chambers, but at the time of installation it was not felt that such construction would be advisable.

The figures show, however, that the plant as constructed paid for itself in less than a year's time, on the basis of pre-war prices for copper. At present prices the investment could be recovered in considerably less time.

Another factor which could now possibly be considered is that of sulphur recovery. A sample analysis of the dust recovered from the drier plant showed over 8 percent sulphur and that from the converter plant over 13 percent. This, however, would doubtless have entailed the construction of an additional treater, operating at a lower temperature, for the sulphur recovery.

The recovery of sulphur, antimony, and arsenic is closely related to the recovery of metallic dust and they may be obtained at the same plants through selective precipitation, the metals being recovered at temperatures above the condensing or dew point of the sulphur, antimony, and arsenic.

There has been an urgent demand for sulphur in the West for the manufacture of sulphuric acid, this demand being considerably in excess of the available supply. The copper leaching plants are the chief consumers of sulphuric acid in the Western field, its use in the dilute form for the leaching of low-grade copper ores for electrolytic extraction having become quite extensive.

A notable instance of the application of electrostatic precipitation as an important step in the process of preparing a suitable acid-leaching solution, is that of the new Cornelia Copper Co. at Ajo, Arizona, where low-grade ores running 2 to 3 percent metal and containing as high as 30 percent sulphur are roasted primarily for the collection of the sulphur dioxide gas, which is used in the treatment of the solution employed at the main plant for leaching a low grade of copper oxide ore. The fume carrying the sulphur dioxide gas is cleaned in the electrostatic precipitator, the analysis of recovered dust showing the presence of As, S, Fe, Al, and Zn.

The first commercial treater for the direct collection of sulphuric acid was that installed at the plant of the Selby Smelting and Lead Co., San Francisco, in 1906, the primary purpose of this collection being the elimination of a nuisance rather than for any monetary value which the recovered material might have. This installation is fully described by Mr. F. G. Cottrell in the Smithsonian Report for 1913, and it is interesting to note that the sulphuric acid solution recovered was employed in the manufacture of copper sulphate, its value being sufficient to pay all operating costs of the plant many times over.

A treater for the recovery of arsenic on a commercial scale is

now under construction as a part of the electrical precipitating plant of the Anaconda Copper Mining Co., at Anaconda, Montana. It has been estimated that 35 tons of arsenious oxide, 92 to 95 percent pure, will be recovered per twenty-four hours.

Potash Recovery.

One of the first applications of electrostatic precipitation on a commercial scale was that at the plant of the Riverside (Cal.) Portland Cement Co., on which work was started in 1911. This was for the purpose of abating the dust nuisance only. However, it developed later, as now well known, that the dust had a very material value in the potash which it contained, and thus afforded another illustration of a peculiar but not uncommon case, wherein later developments of a new process disclosed still other developments which were ranked in importance with that of the original. In the cement industry it has even been demonstrated that the value of the by-product, potash, recovered from cement dust, has a greater value under recent market conditions than the main product itself. The importance of potash as a commercial proposition is perhaps best illustrated by reference to figures on consumption. Mr. John J. Porter, of the Security Cement and Lime Co., has made a considerable study of the matter, and estimates that our pre-war consumption of potash salts were one million tons per year. This is equivalent to approximately 300,000 tons of pure potash, worth about $18,000,000. Over 90 percent of the normal consumption is used as fertilizer.

The average potash content of the limestone which is used in cement manufacture is approximately ¾ percent for the whole country. On this basis, and also on the assumption that 50 percent of the potash is liberated from the raw mixture in the kiln, and recovery of 80 percent of the liberated potash in the precipitator, it is estimated that the cement mills of this country (manufacturing cement from potash-bearing rock) could produce 100,000 tons of potash per year, or approximately one-third of our normal requirements.

As the amount of investment required, the return on same, and time required to recover the investment, are all factors of prime importance, the desirability of instituting such a business is not justified without giving these factors careful consideration. The

attractiveness of the precipitation process from a financial standpoint is again well illustrated by figures for a cement plant operating under average conditions.

It is considered that the precipitated matter should average 8 to 10 percent of water-soluble potash, to be acceptable to the fertilizer industry under conditions of competition with the imported products of a pre-war standard.

Mr. Porter estimates the cost of construction, operation and revenue for an average plant as follows:

With 3,000 barrels (600 tons) per day capacity, 0.75 percent potash in raw mixture, 70 percent liberation when using 1 percent salt, 10 percent ash in coal, and 90 pounds coal per barrel of clinker, 0.9 pound potash recombined per barrel of clinker, 90 percent collection in treater. The conditions prevailing in such a plant are:

Powdered coal burned as fuel. 1 percent of salt added to raw mix to increase liberation of potash in kilns. Selective type precipitation using water-film type treater. Treatment of slurry for liberation of recombined potash, and recovery of pure potash salts by crystallization. This potash would be in the form of relatively pure crystallized salts and there would be no question as to its salability.

"Operating costs and profits would work out about as follows:

Operating Costs

Labor	$ 27.00 per day =	$0.0090 per bbl. clinker (1/5 ton)
Power	25.00 per day =	0.0083 per bbl. clinker (1/5 ton)
Repairs	8.00 per day =	0.0027 per bbl. clinker (1/5 ton)
Laboratory	120.00 per mo. =	0.0014 per bbl. clinker (1/5 ton)
Salt addition		0.0220 per bbl. clinker (1/5 ton)
Total		$0.0434 per bbl. clinker (1/5 ton)

	Present Price	Assumed After-War Price
Sale price per unit of potash	$4.50	$1.00
Value of potash collected per bbl. of clinker	0.4675	0.1039
Operating cost per bbl. clinker	0.0434	0.0434
Operating profit per bbl. clinker	$0.4241	$0.0605
Operating profit per year based on full operation of cement plant	$458,038	$65,340

"This result does not allow for depreciation, royalty, or loss due to part-time operation of cement plant."

Allowing $150,000 for plant cost, which is considered to be a fair figure, investment would be recovered in less than four months. Assuming the after-war price of $1.00 per unit of potash instead of $4.50, the investment would be recovered in approximately two and one-third years. The foregoing figures do not include allowances for depreciation, royalty or loss due to part-time operation.

Figures furnished by Mr. W. A. Schmidt in 1916 on the plant of the Riverside Portland Cement Co. are as follows:

Riverside Installation Cost Balance

Main Treater Installation—			
Platform and housing			$90,000
Treaters and accessories			60,000
Electrical equipment			15,000
Total main installation			$165,000
Multiple pipe treater		$12,000	
Fan, foundations and accessories		3,000	
Total			$15,000
			$180,000
Recoveries—			
Dust, 65 tons at $1.00 per ton		$ 65.00	
Potash material, 20 tons, average 10% K_2O, at 70 cents per unit		140.00	
Recoveries per day		$205.00	
Total per annum			$74,825
Expenses—			
Power, per day	$7.50		
Labor, per day	15.50		
Total, per day	$23.00		
Per annum		$8,395	
Supplies, repairs and accessories per annum		2,000	
			10,395
Gross surplus per annum			$64,430

"The sum of $64,430 covers interest, amortization and profit. Recoveries are based upon full plant operation, but take into consideration the recently reduced dusting of kilns. Potash price is taken at normal market figure (the present market price is $3.00 per unit). All figures are given in average round numbers, as daily fluctuations are large."

"In comparing these figures, it must be considered that the purpose of this installation was to overcome a nuisance, without any consideration of profit. Also this is essentially a dust plant, wherein the recovery of potash is merely incidental. It will be seen, however, that the plant is a sound financial investment, ignoring entirely the question of war-market prices. (Also at the comparatively high cost of installation for a cement treater.)"

The recovery of potash by electrostatic precipitation should not be confined to that from cement mills treating potash-bearing limestones. As a further means of recovery of this valuable and necessary material the precipitation of dust from ferro-manganese furnaces and from iron blast furnace fumes, and of dust from cement mills burning blast furnace slag, is also suggested.

In 1917 there was imported into the U. S. A. approximately 630,000 tons of manganese ore. The United States Geological Survey estimates the local production of high-grade ore at approximately 123,000 tons, making a total of 753,000 tons. A large part of the imported ore comes from Brazil, and according to analyses given by R. J. Wysor, Supt. of Blast Furnaces of the Bethlehem Steel Co., in his paper presented before Am. Institute of Mining Engineers in Feb., 1917, the potash content averages 0.88 percent. This would appear to represent another valuable source of potash, which might be recovered by electrostatic precipitation as a by-product of the manufacture of ferro-manganese in the blast furnace, wherein the dust losses are said to run approximately 25 percent. There has also been increased activity in the manufacture of ferro-manganese in the electric furnace from low-grade ores mined in the United States. With regard to recovery of potash from blast furnace dust, Mr. Wysor, in giving the results of his extensive investigation, says:

"The percentage of K_2O in the ores investigated averaged 0.28; the percentage K_2O in the Lehigh by-product coke from West Virginia coals employed, averaged 0.28; the percentage of K_2O in the flux employed (½ McAfee limestone from northern New Jersey and ½ average dolomite from Bethlehem vicinity), averaged 0.59. Since a furnace charge consisted of approximately 3,700 lb. (1,670 kg.) of ore, 2,070 lb. (912 kg.) of coke and 1,070 lb. (486 kg.) of flux for each ton of pig iron produced, there was 22.4 lb. (10 kg.) of K_2O introduced into the blast furnace for

each ton of pig iron produced. Of this 22.4 lb. (10 kg.), 4½ lb. (2.04 kg.), that is, 20 percent, went into the slag and would be recoverable by the precipitation process from the stacks of the cement plants working on cement from the blast furnace slag.

Since the pig iron production in the United States in 1916 amounted to 39,000,000 tons, we determine by assuming the figure of 22.4 lb. (10 kg.) to be a fair average, that a total of 400,000 tons of K_2O was introduced into the blast furnaces during that year, and, further, assuming the figure 4½ lb. (2.04 kg.) to be a fair average, that 80,000 tons of K_2O went into blast furnace slag. In the discussion of Mr. Wysor's paper it was brought out by Mr. J. S. Unger, of Pittsburgh, that, in blast furnaces with which he was connected, operating principally on Lake Superior ores, Ohio and Pennsylvania limestone and Connellsville coke, he found about ½ percent K_2O in the furnace slag. Mr. Wysor stated that at Bethlehem the percentage of K_2O in the slag was something less than 0.4. Mr. Wysor explained the difference by stating that he had determined that Connellsville coke has a lower alkali content than by-product coke made from West Virginia coals, and he believed there was a difference in the practice of operating the blast furnaces in that the Western blast furnaces operated with lower hot-blast and top temperatures than at Bethlehem. After all, there is not a great difference between Mr. Wysor's and Mr. Unger's experiences, and we may assume from 8 to 10 lb. (3.6 to 4.5 kg.) of K_2O in each ton of blast furnace slag.

According to the balance sheet for alkalies charged, removed, and lost, in Mr. Wysor's paper, the total K_2O charge was 22.4 lb. (10 kg.) or 100 percent, and only 4 percent was recovered. If this figure applied for the whole country in 1916, then only 16,000 tons of the 400,000 tons total of K_2O charged into American blast furnaces were recovered.

According to Mr. H. D. Egbert, of the Research Corporation, in his article "Electrical Cleaning of Blast Furnace Gas," published in the January, 1918, issue of "The Blast Furnace and Steel Plant," 71.5 percent of the total K_2O charged into the furnace is recoverable by a hot dry-cleaning process. Assuming that the 20 percent going into the slag is also largely recoverable, then we may assume that roughly, 85 to 90 percent of the total K_2O charged into the blast furnace is recoverable by the electrical precipitation process.

The possible recovery then from this source, combined with that from cement mills previously mentioned, would be considerably in excess of our normal requirements.

In this connection it is interesting to note the amount of potash actually being recovered. The output of all potash materials produced and marketed, as reported by manufacturers to the U. S. Geological Survey in 1917, was 126,577 tons, which averaged 26.4 percent, or 32,366 tons, of pure potash. The total value was $13,791,922. Of the total potash material produced, approximately 10 percent was recovered from cement mills, about half of which employed the electrostatic precipitation process.

Treatment of Emulsions.

The treatment of emulsions by the electrical precipitation process, for removal of suspended material, has received considerable impetus, chiefly in the dehydration of oils in California oil fields and to a limited extent in the removal of tarry matter from gases.

In the practical production and transportation of crude petroleum, water is often introduced in the oil. This water may be in the form of large globules which will settle out if the mixture is allowed to stand, such water being known as "free water." Water may also be present in very minute particles forming an emulsion with the oil, which will not separate into its component parts even if allowed to stand for long periods of time. The water in such emulsions has been commonly described as "trapped water." The removal of the "trapped water" has been the object of numerous patents which have been taken out covering this phase of the process, with varying degrees of success.

Practically all patents are based on the fundamental idea of introducing the emulsion into an electrostatic field produced between electrodes on which an alternating e. m. f. is impressed, the minute particles of water being coalesced, forming drops of free water, which may be readily removed.

It is claimed that emulsions running as high as 65 percent water have been dehydrated to 0.5 percent water and refuse, and that where power can be purchased for 1.5 cent per K.W.H. this process is far more economical than evaporation by use of steam.

The process seems to be admirably adapted for this service,

especially in regard to its effectiveness, simplicity and cheapness, which are all factors of prime importance. With the enormous increase in oil production due to the stimulus of the war, the conditions for further extensive development of the process in this field appear very promising, and it is to be hoped that the fullest realization of the benefits and the maximum possible advancement in the art will be attained.

While some work has been done towards perfecting a practical process for removing tarry matter from gases, *viz.*, that conducted at the research laboratory of the University of Michigan under direction of Mr. A. H. White, in 1914, it does not seem to have had a very wide commercial application.

The process has been described by Mr. F. W. Steere in a paper read before the American Gas Institute in 1914, entitled "An Electrical Process for De-tarring Gas," and consists essentially of the agglomeration of extremely fine particles of tar mist into larger particles by means of an alternating current electrostatic field, the agglomerated particles being of a size which may be removed by means of a mechanical device, such as a centrifugal extractor. The process was intended primarily for the purpose of removing the tar as an undesirable element in the gas, the combination of tar and naphthalene forming a sticky material which seriously interferes with the operation of gas-producing apparatus. Aside from the removal of tar, other advantages were obtained, *viz.*, an increase in the amount of light oil left in the gas, the recovery of naphthalene, and very pure ammonium sulphates. The chief application of this process on a large commercial scale would seem to be in that of the by-product coke industry, although it has not been used extensively in such plants installed in the last few years.

It is suggested that a very promising application of the precipitation process would be found in the carbon baking plants (where baking is done by producer gas), for the purpose of clarifying the heavy fumes which otherwise may constitute a nuisance due to their destructive effect on vegetation. While it is not expected that this would reach great commercial proportions it would nevertheless have a high intrinsic value, and if carried to a successful commercial development would stand as another example of a simple and efficacious remedy for a perplexing problem.

The collection of carbon dust from calcium carbide furnaces has been accomplished by means of electrostatic precipitation, and it is to be expected that this application will receive considerable development because of the increasing number of such furnaces which are being placed in operation.

It is difficult to enumerate all of the uses of the electrostatic precipitation process, as many applications have been made in comparatively small capacities, of which there is little generally known. In order to keep before us the proper conception of the scope of this work, there may be mentioned, in addition to the foregoing, the following applications: Cleaning of air drawn from buildings in which dust-producing operations are carried on, removal of soot from smoke given off in railway roundhouses, recovery of phosphoric acid from gases given off from furnaces treating phosphate rock, recovery of zinc oxide fumes from brass foundries, and purification of noxious or poisonous gases.

Treaters.

As may be reasonably expected, there is no one type of treater or precipitating chamber which is universally adapted for the treatment of materials. The commercial treaters, as at present developed, may be classified according to service and type as:

1. *Gas treaters,* of which there are three general types:
   (a) *Tube.*
   (b) *Plate.*
   (c) *Plate type employing water film.*
2. *Liquid treaters,* employing:
   (a) *Stationary electrodes.*
   (b) *Rotating electrodes.*

By far the greatest application of the electrical precipitating process has been to the treatment of gases for the recovery of the suspended particles. Of the precipitators for the treatment of gas, the tube and plate type have been most generally employed, the use of the former predominating in most installations.

The tube type of treater consists of two enclosed compartments connected by a nest of metallic, cylindrical tubes mounted in a vertical position, with a wire or chain suspended centrally in the tube, the tube acting as the receiving electrode, and the

wire or chain as the discharge electrode. This type of treater as applied to the treatment of hot dry gases from a cement mill is shown in Fig. 1.

The compartments serve to provide a means of connecting the flue carrying the gases to the precipitating tubes, and in such a manner as to cause an even distribution. Various features have

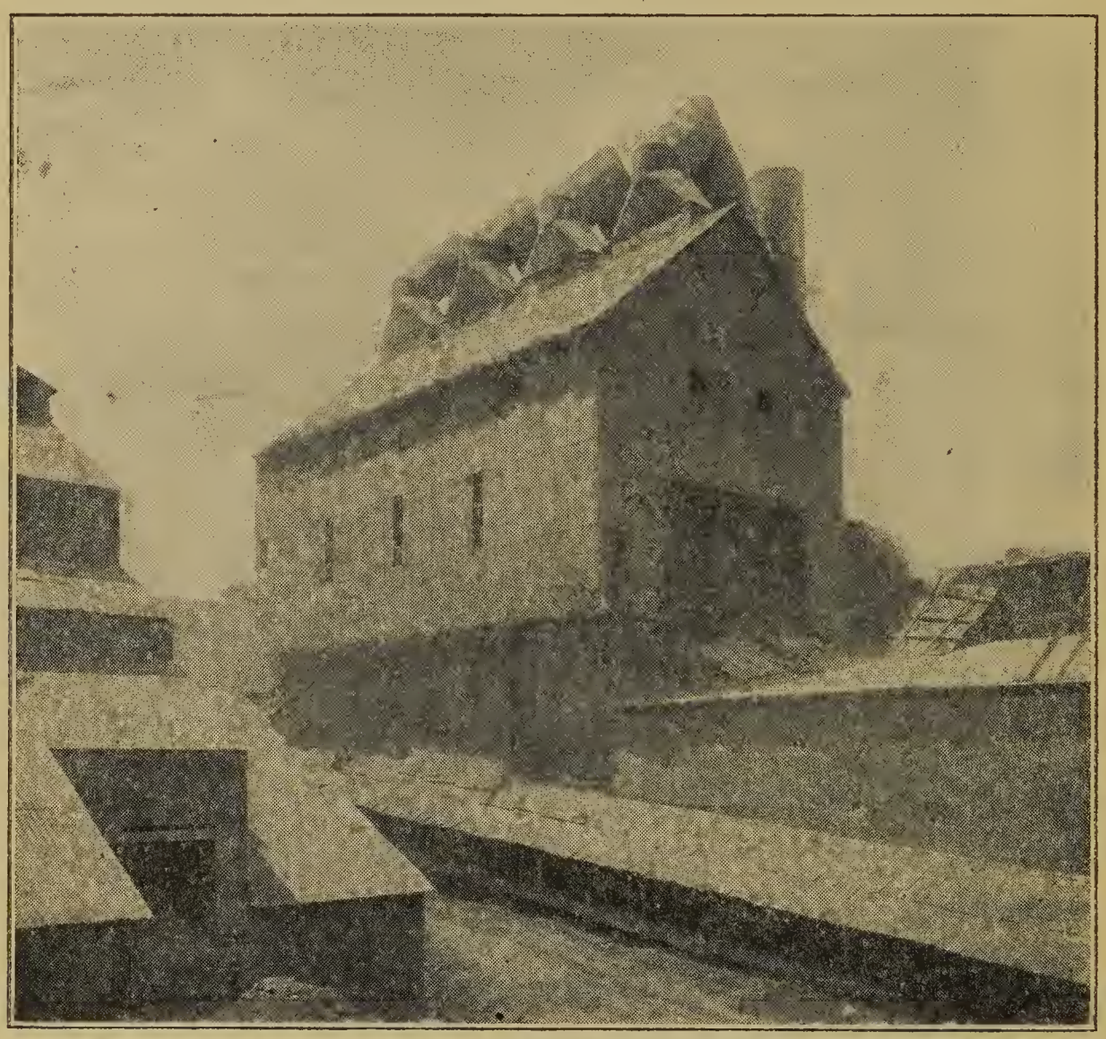


FIG. 1. Installation of tube type dry treater for the precipitation of cement dust at the plant of the Security Cement and Lime Co., Hagerstown, Md. The fume enters treater through flues at the top and passes downward through the treater.

been incorporated by way of refinement of design to provide for the successful treatment of the various types of gases. Among these may be mentioned: housing around the tubes and flues for temperature control, sectionalizing of treaters for the purpose of maintaining continuous operation of process during period requiring parts of treater to be cut out for repair or cleaning, dampers and baffling plates for controlling the volume and distribution

of gas to individual treater sections, shrouding of discharge electrodes at points of entering tubes to prevent arc-over at these points due to the intense electrostatic field at end of tubes, providing tension on discharge electrodes by means of weights or springs to center them and to prevent vibration, special high voltage insulator construction to withstand service under conditions of dust and wide temperature variation, rapping mechanisms for removing deposited material from electrodes, special materials for electrodes depending on nature of gas being treated, etc., etc.

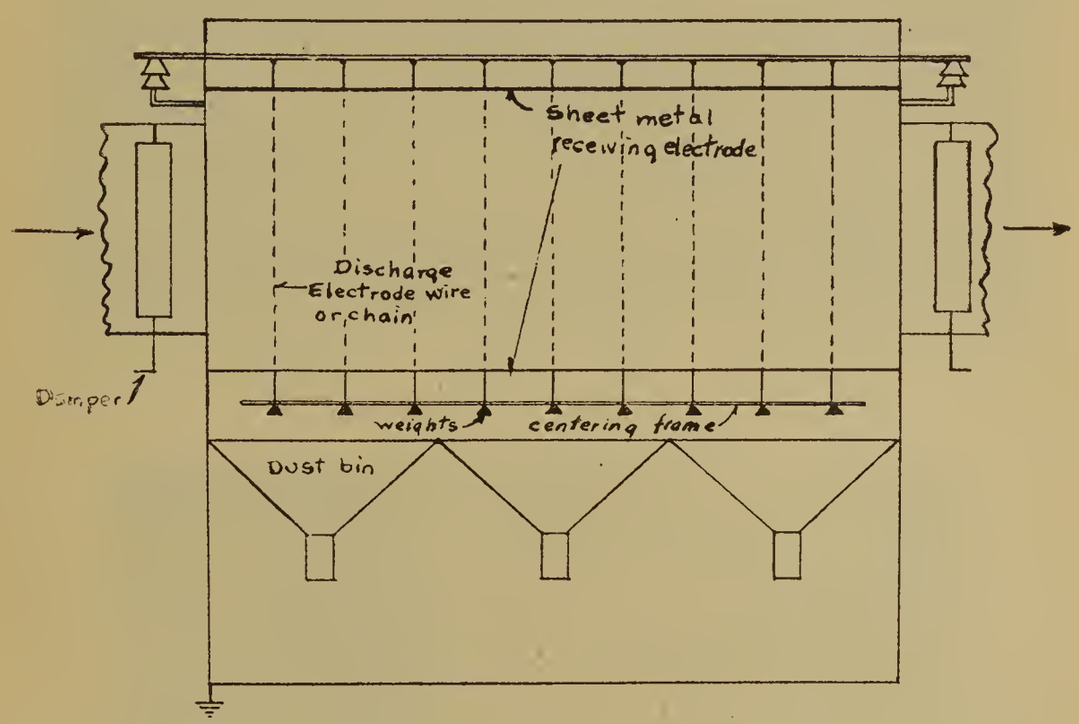


Fig. 2. Sketch of a plate type precipitator for fume treatment. Side elevation.

The plate type of treater as adapted for gas, illustrated by Fig. 2, consists of a number of sheet metal plates acting as receiving electrodes and suspended in a compartment, the discharge electrodes consisting of wires or screens, suspended equidistant between the plates. This type has been applied for the recovery of cement dust, metal bearing fume such as that from lead and copper smelters, and for acid precipitation.

The plate type of water film treater, illustrated by Figs. 3(a) and 3(b) is built on the same principle as the plate type dry treater, except that the plates or receiving electrodes are in the nature of partitions over which a continuous film of water is maintained.

FIG. 3(a). General view of water film treater at the plant of the Security Cement and Lime Co., Hagerstown, Md. Discharge is at the top.

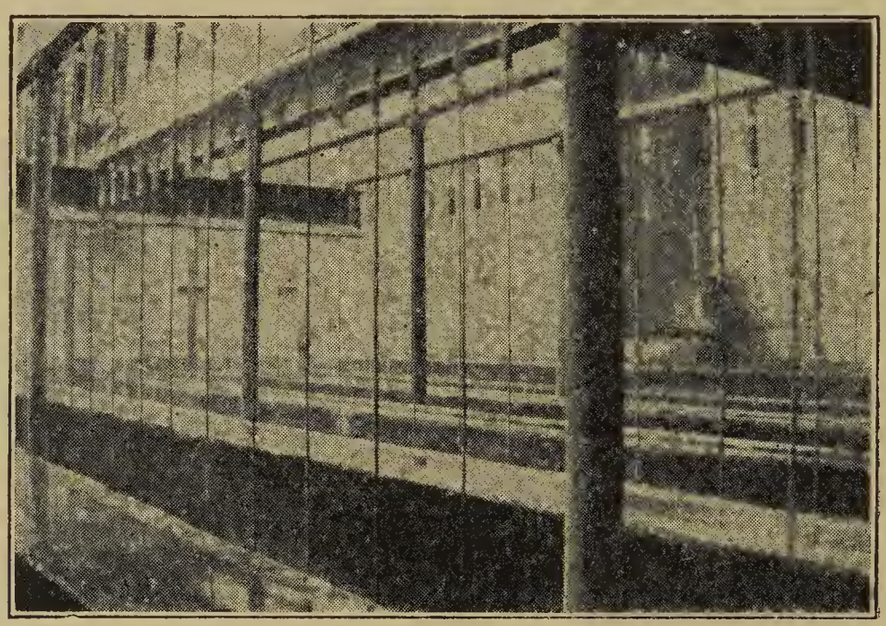


FIG. 3(b). Interior view of one section of water film type treater shown in Fig. 3(a). Discharge electrodes are No. 14 wire held taut by spring tension. The concrete receiving electrodes are covered with a water film—the water overflowing from a trough in the top of the electrodes.

This type of treater has recently been applied to the recovery of potash from cement mill dust, in connection with the dry type treater, the latter collecting the coarse dry dust at a comparatively

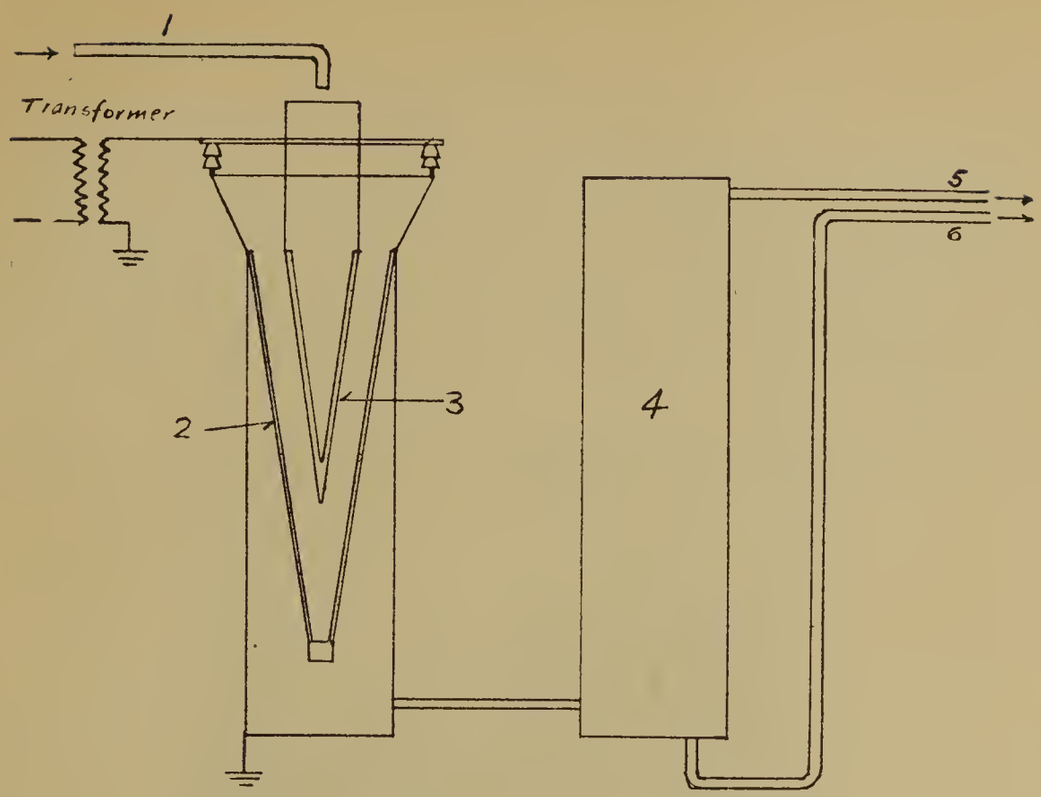


FIG. 4. Sketch of an emulsion treater with fixed electrodes, wherein the electrodes consist of cone-shaped filter screens. The emulsion enters at 1, water is coalesced between electrodes 2 and 3 and runs to bottom. The separated constituents flow to settling chamber 4, the oil and water being withdrawn at 5 and 6 respectively.

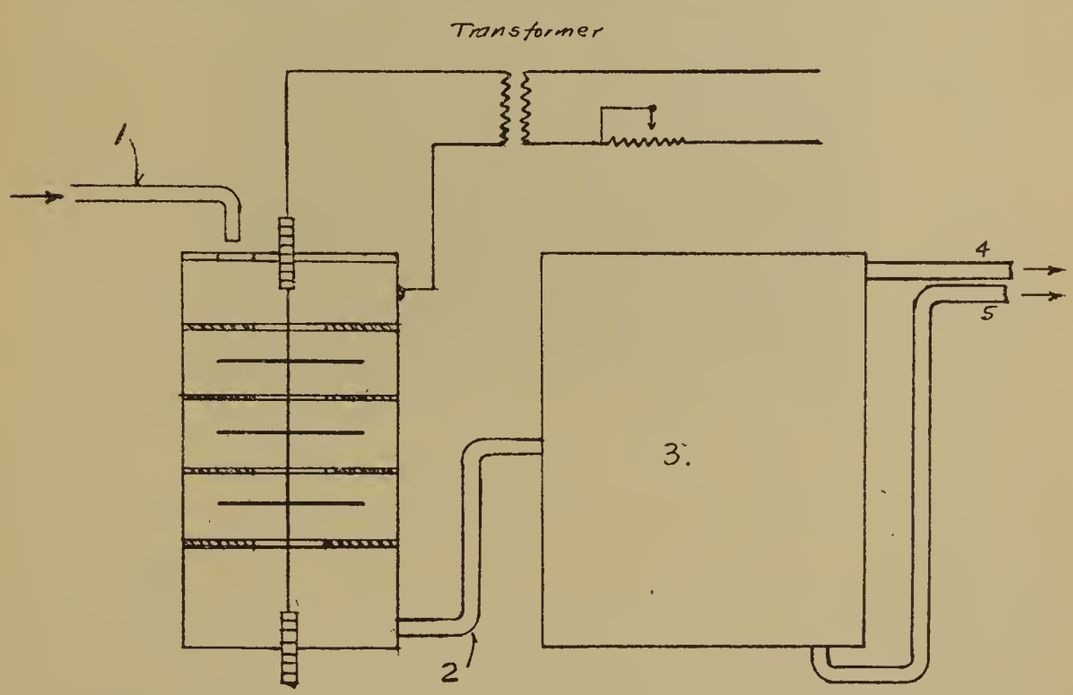


FIG. 5. Sketch of an emulsion treater with fixed electrodes and settling chamber at right. Emulsion enters at 1, the oil and water being withdrawn separately at 4 and 5.

high temperature, the former collecting the remainder of the dust in solution, thus eliminating the trouble due to the presence of chlorine or chlorides and obtaining a soluble potash which may

be crystallized out into the form of highly concentrated potash salts.

The treaters for dehydration of emulsions have been made in various types, of which Figs. 4 and 5 are examples. It is interesting to note that some of the patents claim that means are provided for the *prevention* of the formation of complete chains of coalesced particles of water, which would short-circuit the electrodes, while others claim that formation of such chains is *essential* to the success of the process, and this feature is incorporated in the patent. Fig. 4 is an example of the type designed to prevent short-circuiting of the electrodes, while Fig. 5 is that in which short-circuit is supposed to occur. Fig. 6 also shows a type similar to the latter, in the California oil fields. Fig. 7 illustrates diagrammatically the treater apparatus described by Mr. F. W. Steere for the removal of tarry matter from gases.

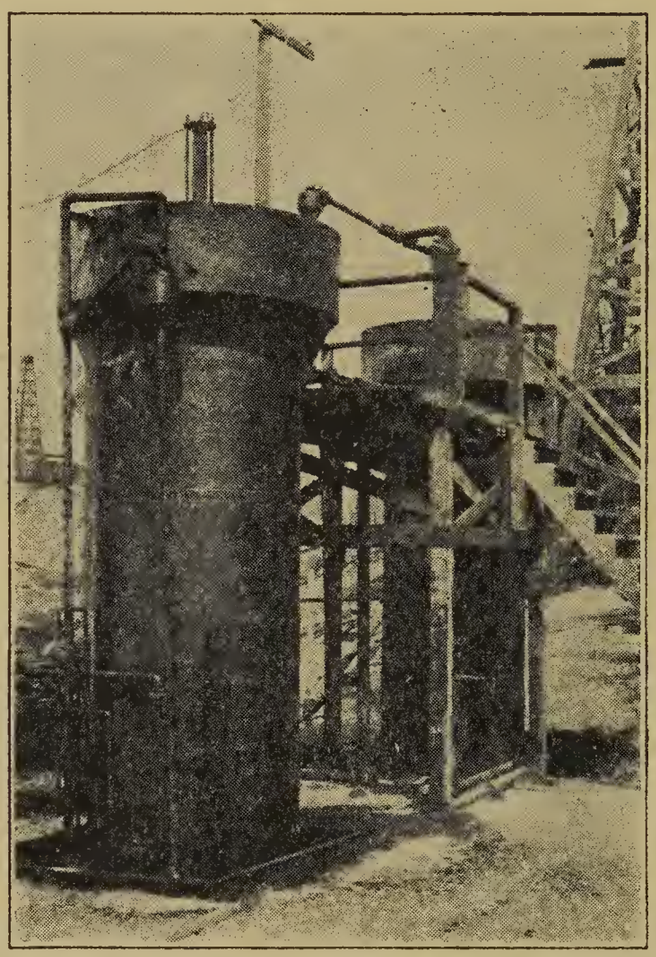


Fig. 6. Installation of an electrostatic precipitator in the California oil fields for the dehydration of crude petroleum oil.

Temperature.

Temperature control of the medium being treated is recognized as being one of the important factors essential to the success of the process. No definite temperature can be given which will apply to all classes of mediums treated, as there are other factors, for example, the velocity and characteristics of fume being treated, in themselves variable quantities, which influence the operation. Selective precipitation of the different constituents of gases is accomplished by maintaining a temperature in the

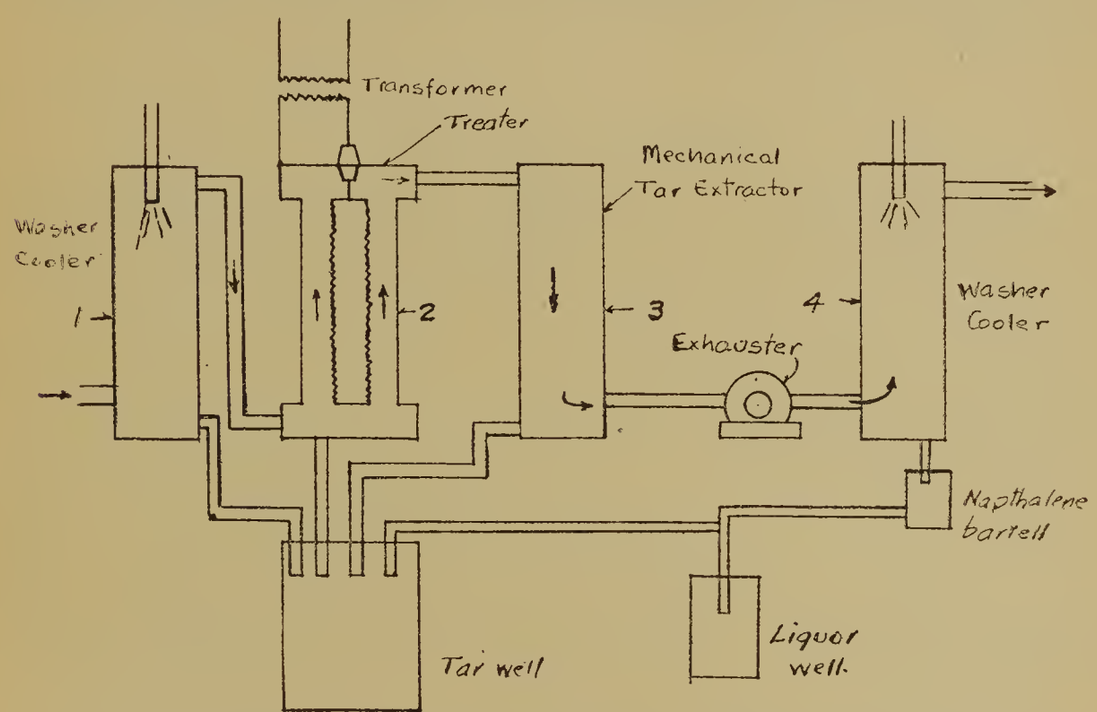


FIG. 7. Sketch of apparatus used for the extraction of tar from producer gas. The gas enters the chamber (1) at the left, the tar mist is agglomerated in the treaters (2) and then collected by the centrifugal extractor (3). From sketch in article by Mr. F. W. Steere, Proceedings of the American Gas Institute, 1914.

treater which establishes the most favorable conditions. Thus, in the treatment of fume from copper smelters the metallic dust is first precipitated at high temperature, and then by cooling the gas to the condensing point of arsenic, the latter may be recovered.

In reviewing the operating practice prevailing at a number of plants treating various materials, it has been observed that the following temperatures represent a fair average: In cement mills the temperature of dry treaters 300° to 500° C., wet or water film treater 80° to 100° C.; copper smelting plants, metallic dust treaters 180° to 200° C.; arsenic treater, below 125° C., lead smelting plants, metallic dust, 80° to 120° C.

Gas Velocity in Treaters.

The gas velocity employed in a majority of treaters averages 5 to 6 ft. (1.5 to 1.8 m.) per second—some installations operating as low as 2.5 ft. (0.75 m.) second, and others as high as 8 to 10 ft. (2.4 to 3 m.). Some of the smaller plants operating on natural draft are of course not susceptible to such control as are those on forced draft. The fan for supplying forced draft has in some instances been placed ahead of the treater and in others behind, the gases passing through the tubes under pressure or suction respectively.

Direction of Gas Flow.

In tube-type treaters the direction of gas flow is vertical, and plants have been installed for both upward and downward draft. In general, it is perhaps more convenient from a construction standpoint to provide an up-draft, the treater in some ore-drying plants being placed above the sinterers and roasters. Down-draft perhaps gives a better gas distribution in the treater pipes than up-draft. The plate type of treater has been constructed for the passage of gas in both horizontal and vertical directions. This type of construction is being installed in the new plant of the Anaconda Copper Co., for handling an immense volume of gas.

Capacity of Electrical Equipment.

In reviewing the operating practice of a number of installations it is observed that there is quite a wide variation in the capacity of electrical equipment for the amount of gas being treated. That there would be considerable variation is to be expected, since the factors which affect it may also vary, and furthermore some plants have more surplus capacity than others. It may be noted, as a matter of general interest, that in two cement plants the volt-amperes per 1,000 cu. ft. (29 cu. m.) gas per minute averages 175, while the average of 16 copper and lead fume treaters is 600. Similarly the average volt-amperes per square foot of electrode surface is 1.25 (14 per sq. m.) for the cement plants and 3.00 (33 per sq. m.) for the copper and lead fume treaters.

Operating Voltage.

The voltage which may be maintained on the treater for a given velocity of discharge is influenced and determined by the temperature, physical characteristics, electrode dimensions, etc.

The actual voltage obtained on the treater is often overestimated, a common method of estimating being to assume the sine-wave form of the alternating current and to multiply the A.C. voltage, as given by transformer ratio, by $\sqrt{2}$. There are several factors which cause error in this method of calculation. They are:

(1) The actual transformer tap employed is lower than the maximum, which is the value commonly assumed.

(2) The drop in voltage across resistances which are generally connected in either high voltage or low voltage circuits, or both.

(3) Drop in voltage in rectifier.

(4) Flat wave-top form of the voltage, which is a resultant of load characteristics.

Data obtained from a number of installations indicates that the maximum voltage maintained on the various types of treaters is seldom in excess of 60,000 volts.

Electrical Apparatus.

The proper selection of apparatus for the delivery of electrical energy to the treaters is controlled by many of the factors which determine treater design, the two of necessity having requirements in common. However, due to the present-day status of the electrical apparatus which has been developed and standardized for this service, the selection of this equipment has become a comparatively simple matter.

In the treatment of gases for fume, a uni-directional high voltage current for maintaining a high static charge on the electrodes is required, as pointed out previously. The most practicable method developed thus far for providing this is that of rectifying a high-voltage alternating current, the latter being provided by stepping up a low-voltage to a suitable value by means of a transformer especially designed for the purpose.

The rectification is usually accomplished by means of a mechanical switch running in synchronism with the A.C. supply.

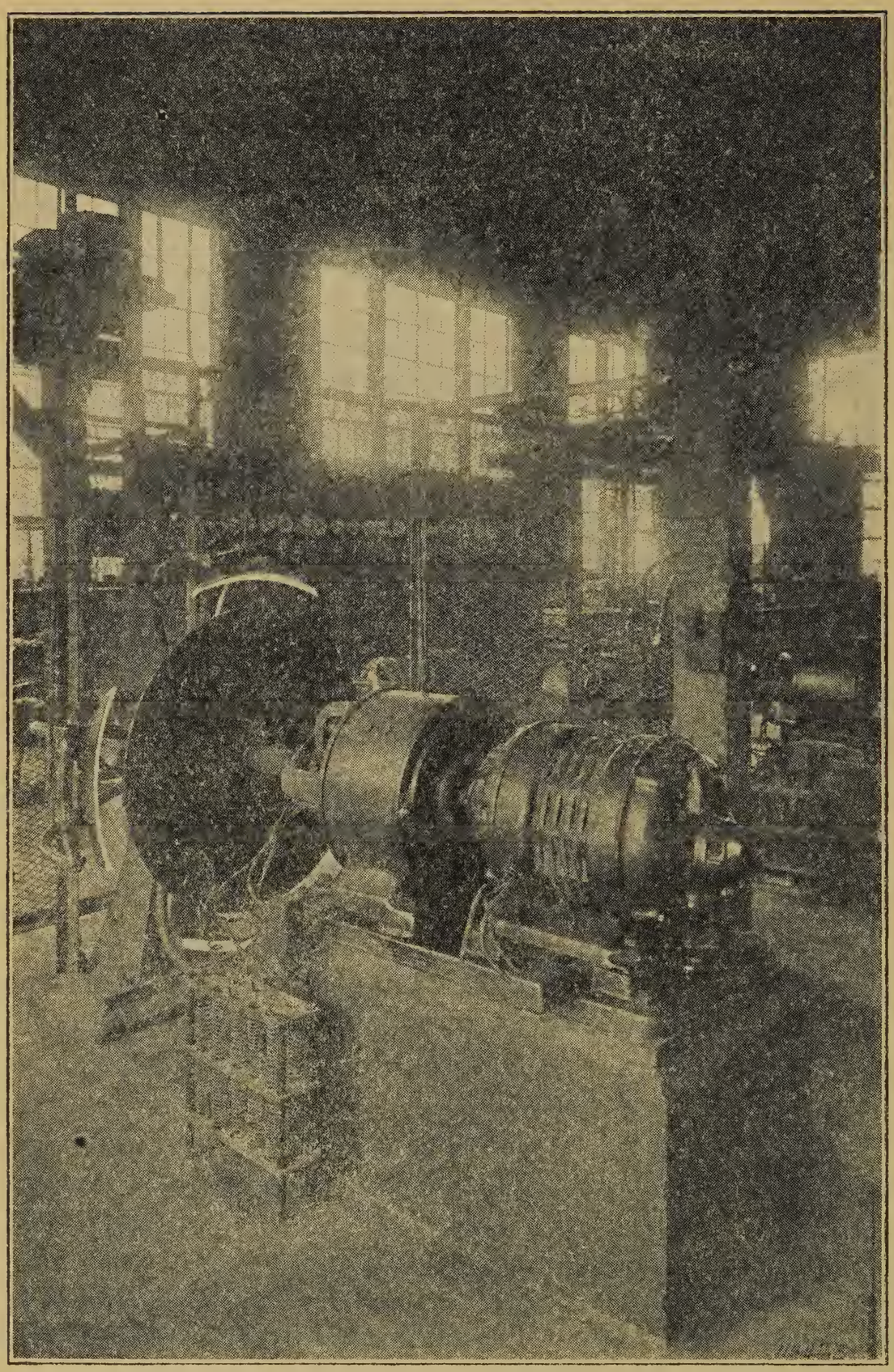


Fig. 8. An installation of a disc type mechanical rectifier direct connected to the shaft of a 15 K.V.A., single phase, alternating current, motor-driven generator.

This type of rectifier, direct connected to a motor generator set, is shown in Fig. 8. Means are usually provided for suppression or smoothing out oscillations set up in the circuit by the rapid interruption of the high-voltage current. This is usually accomplished by the insertion of resistors in either the high-voltage or low-voltage circuits.

From the foregoing very general description, the electrical requirements may be summarized as follows:

(1) A source of A.C. electrical energy.

(2) Means of transforming this energy to a suitable potential.

(3) Means of rectifying the high potential alternating current.

(4) Means of control, including necessary auxiliary apparatus.

There are three general systems in use for supplying energy to the treater, the electrical apparatus required being determined by the system selected. These are:

(*a*) Individual motor-driven generator for each treater unit.

(*b*) Power taken directly from an industrial lighting or power circuit.

(*c*) One large generator of sufficient capacity to supply all treater units.

No particular one of the three methods of obtaining a source of energy can be selected to the exclusion of the others, although it may be said that the first and second arrangements are more generally preferred. The individual motor-driven generator has been used in the larger installations to a greater extent than arrangements (*b*) and (*c*), it being generally recognized that the advantages of this system are:

(1) The treater circuits are entirely independent electrically from the primary source of supply, with the result that disturbances in the treaters are not reflected in the supply circuit, and *vice versa*.

(2) The voltage of the individual treater circuits may be regulated very simply and closely by means of generator field excitation, and furthermore is independent of moderate variations in the supply voltage.

In the second arrangement, wherein the step-up transformer is connected directly to the source of supply, and the voltage on

the treaters may be controlled by means of an adjustable resistor, placed in the primary low-voltage circuit, a small synchronous motor is employed to drive the rectifier switch in synchronism with the supply voltage.

The advantages claimed for this arrangement over that first described are:

(1) Less first cost due to substitution of a small synchronous motor for a motor generator set.

(2) Less floor space.

While the first cost is less, this item is not of great importance, since it has been demonstrated that the cost of the electrical equipment is often only 10 to 15 percent of the total cost of the complete installation. It is also recognized that with the second arrangement it is necessary that the supply circuit be of large capacity as compared to the treater load, or of close voltage regulation, otherwise treater disturbances may be reflected back to the supply system causing disturbances which may set up oscillations in other treaters operating from the same circuit, or set up fluctuations in line voltage which may seriously interfere with successful treater operation.

The third arrangement is a combination of the first and second and the foregoing remarks for the two may be here applied, with proper consideration to the advantages and disadvantages of each.

Transformers.

It is perhaps not too much to say that the development of the high-voltage transformer has been the most important apparatus factor in placing the electrostatic precipitation process on a practical commercial basis. It was largely lack of sufficient capacity at high potential which proved a serious stumbling block to the early experimenters.

The unusually severe requirements of the service demand that the transformer be designed to withstand high static strains within the windings, and the use of special high-voltage terminals. Other requirements are: low exciting current, as low reactance as consistent with insulation requirements, and special taps to provide a means of voltage adjustment.

Generators.

The A.C., single-phase, synchronous, low-voltage generators employed in class "A" service for supplying power to treater currents, are provided with special characteristics to conform to the requirements of the load. Such machines are separately excited, and provision is made for a wide variation in voltage by means of field control, generally from 50 to 125 percent of normal. It was generally accepted in the past that the treater load was of leading power factor—specifications for generating apparatus specifying from 90 to 70 percent leading P.F. Later developments have shown, however, that the P.F. of the treater circuit is often nearly 100 percent and sometimes as low as 90 percent lagging.

While the construction of the treater itself is essentially that of a condenser, the actual capacity in microfarads, even in large installations, is quite small. The dissipation of true energy may take place in the treater tube itself in the deposition of particles, in leakage over insulators and H.V. wiring, and losses in resistors and transforming apparatus. The effect of reactance in the transformer, generator, and choke coils also tends to offset the leading power factor of the treater.

The items contributing to the successful operation of these generators are:

(1) Liberal design of fields, to provide stability and close regulation.

(2) Characteristics designed to give a smooth voltage wave, and for damping out oscillations which may have passed through the step-up transformers.

(3) Drooping voltage characteristics on severe overloads, for protection of generator and transformer in case of short-circuit in treater tubes.

(4) Liberal design of mechanical parts, such as bearings, collector rings, and brushes, which are often required to operate under very adverse conditions as regards dust and dirt.

Synchronous Motors.

The small synchronous polyphase motors employed for driving the mechanical rectifier switch, in installations taking power from

an industrial circuit, may be built as either straight synchronous motors, or, as more generally done, of the induction type, the rotor having salient poles with a squirrel cage starting winding. The synchronous induction type requires no source of separate excitation or special means for starting, and is, therefore, generally preferred. The capacity required to drive the various types of rectifier switches is seldom in excess of two H.P. A type of this motor direct connected to and driving a disc type rectifier is shown in Fig. 9.

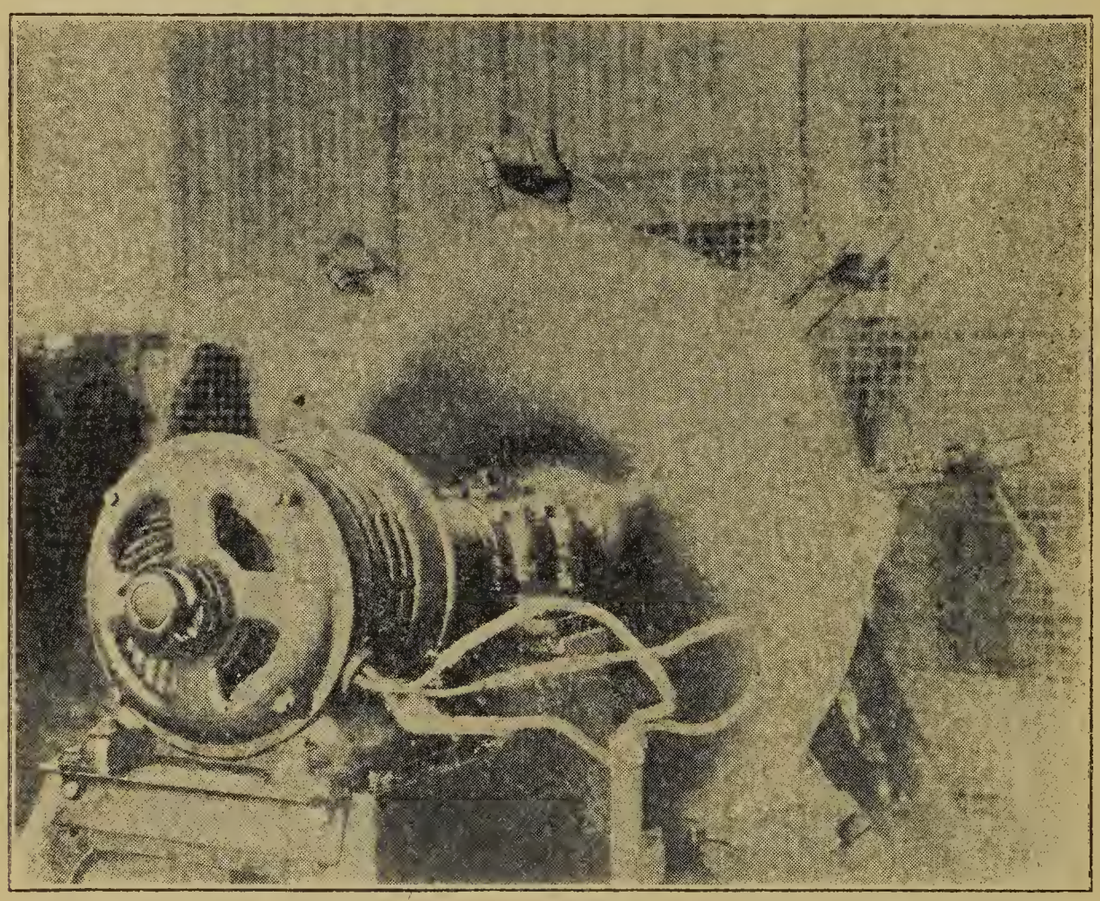


Fig. 9. Synchronous motor driving a mechanical rectifier switch. Motor is rated at 2 H.P., 220 volts, 3 phase, 60 cycles, 1,800 R.P.M.

Rectifiers.

Various means have been devised for the production of direct or unidirectional current at high potential, among which may be mentioned: the rotating mechanical rectifier switch, hot cathode converter or kenotron, and high-voltage direct current generator.

The requirements of the apparatus for any successful commercial treater installation demand:

(1) Ability to operate at conditions productive of the maximum recovery in the treater.

(2) Uninterrupted service continuously maintained under above conditions.

(3) Ease of adjustment to changing operating conditions.

(4) Simple, rugged construction, and therefore low maintenance.

(5) Low first cost.

The mechanical rectifier used in conjunction with the high-voltage transformer has been the only apparatus thus far developed which has met all of the above requirements, and it is therefore used almost exclusively in all commercial installations. The successful development of the mechanical rectifier has involved:

(1) Production of a rotating element of light weight, yet with sufficient mechanical strength to permit of operation at high peripheral speed, co-ordinated with high dielectric properties to withstand the static strains incident to commutation of a high-voltage oscillating current.

(2) Provision of means for shifting the position of the stationary contacts (or shoes) with respect to the rotating contacts, thereby affording a means of controlling the portion of the wave which is commutated, and hence the energy input to the treater.

(3) Improvement in design of arcing contacts.

A feature of the mechanical rectifier, which is a distinct asset and yet perhaps not generally appreciated, is the discontinuous characteristic of the rectified wave, which tends to maintain the voltage and suppress short-circuits in the treater. This point has been brought out by Mr. O. H. Eschholz in his paper "Some Electrical Characteristics of Electrostatic Precipitation," read at the September, 1918, meeting of the American Mining Institute, in which it is shown that due to the slow rate at which the large mass of charged particles move, where an arc is not formed, as compared to the rate at which electrical charges are delivered to these particles, the treater tends to maintain a constant voltage and "the rectifier practically functions on this application to convert a high-tension alternating current to a practically constant high-tension direct current." Treater short-circuits or flash-overs

of a local nature are suppressed, since during the period when the circuit is opened by the rectifier switch an opportunity is afforded the treater to discharge with practically no interruption in its operation.

Figs. 10 and 11 show photographs of the arc at one of the commutating points of a mechanical rectifier, operating on a 60-cycle circuit, at approximately 1,800 R.P.M.

Exposure 1/50 second.

Exposure 1 second.

FIGS. 10 AND 11. Photographs of rectifier arc. Rectifier rotating counter-clock-wise.

The high-voltage direct-current generator has only recently been produced, but as yet its ability to meet the operating requirements has not been established by extended trial under the varying conditions met in commercial installations.

Some of the characteristics of such a generator which would seem incompatible with the requirements of the service are:

(1) Poor voltage regulation occasioned by high reactance, high armature resistance, jumping and arcing of the brushes on the commutators, and interruptions to service caused by treater short-circuits. (It may of course be possible to eliminate most of these undesirable characteristics by providing suitable auxiliary regulating and protective apparatus which would, however, be rather elaborate and require delicate adjustment.)

(2) In general, considerable maintenance difficulties and expense may be expected on any machine which has rotating parts revolving at high speed and which must also be insulated for very high voltages.

(3) The construction of both generator and any necessary auxiliary control apparatus would undoubtedly be far from simple and would therefore require constant skilled attendance and involve considerable delay in replacing parts in case of damage.

(4) High first cost.

The hot cathode rectifier, or kenotron tube, is similar to the mercury vapor rectifier in that low resistance is offered to the passage of current in one direction and very high resistance to the passage of current in the opposite direction. The current, however, instead of being conveyed by ionized gas, is carried by a stream of electrons passing from the incandescent filament to the cold electrode through a very high vacuum. Either one or both halves of the alternating wave may be rectified, as desired. While this device has been employed in the rectification of high voltage, its application would seem to be limited to rather small capacities, and its ability to withstand the severe service requirements are questioned. Apparently the most serious limitations are:

(1) High maintenance expense and interruption of service due to

(*a*) Fracture of bulbs by very high-voltage static discharges.

(*b*) Burning out of incandescent filament, which operates close to the melting point and therefore requires very close regulation of the filament current.

(*c*) Rather elaborate and delicately adjusted apparatus for protection against short-circuits. (A short-circuit would generally require a complete replacement of the tubes.)

(2) Multiplicity of small capacity tubes are required to obtain large capacity, and this therefore multiplies the above difficulties.

Switchboard and Accessories.

A schematic outline of circuits for a treater installation is shown in Fig. 12, while general arrangements employed for class *A* and *B* systems are given in Figs. 13 and 14. The switching equipment is usually quite simple, and is deserving of no special attention. Fig. 15 shows a typical switchboard installation for this service.

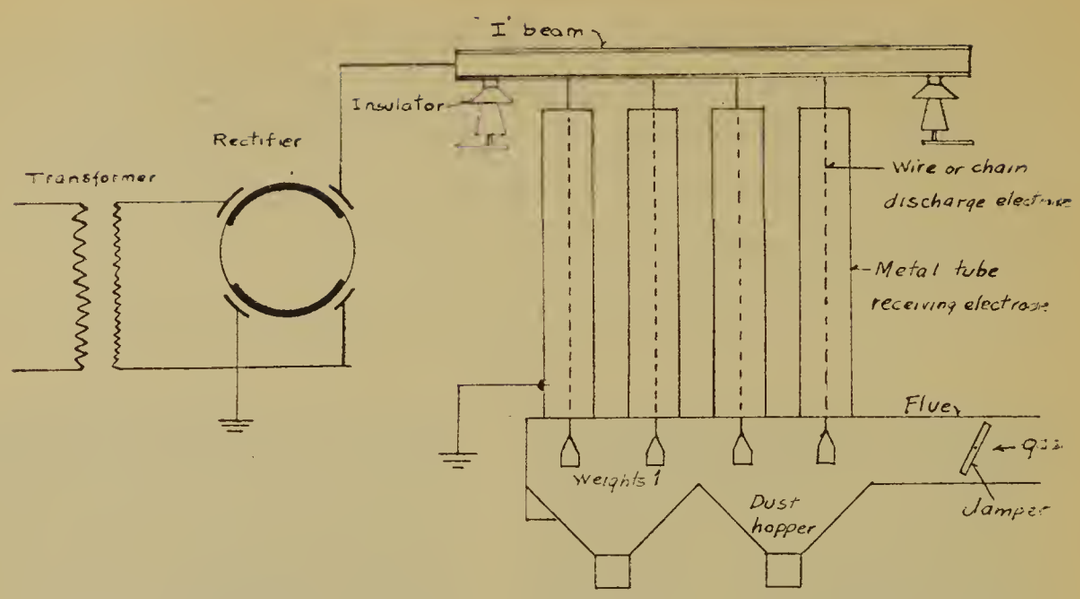


FIG. 12. Schematic diagram, illustrating arrangement of treater circuit.

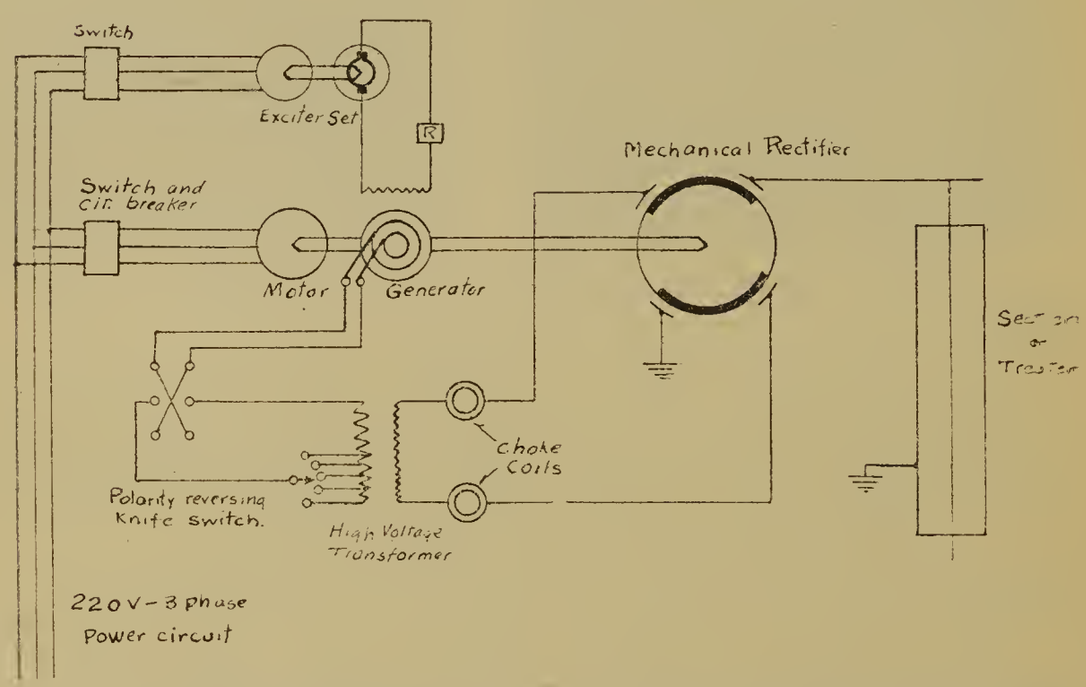


FIG. 13. Typical connections for class "A" service. Each transformer with individual motor-driven single-phase generator.

The protective devices which have been employed to reduce the effect of surges and oscillations of the circuit are quite simple. It is customary to install resistors either in the high-voltage or low-voltage circuit, the resistance generally being 10 percent to 20 percent of the capacity of the circuit. Resistors placed in the high-voltage circuit generally consist of carborundum tubes or similar high-resistivity material, while those placed in the low-voltage circuit may be of the grid or wire type, depending on capacity.

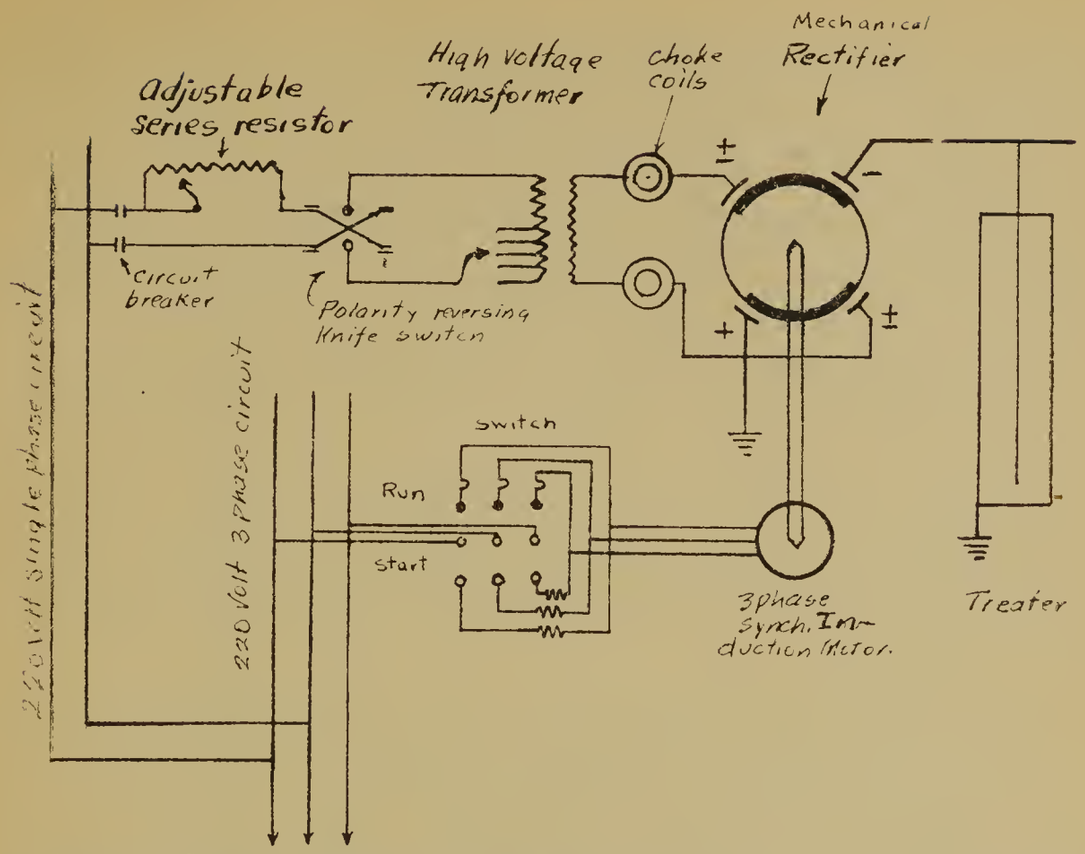


FIG. 14. Schematic diagram, illustrating class "B" service, wherein several transformer-rectifier-synchronous motor sets are operated from an industrial circuit. Diagram shows connections for one unit.

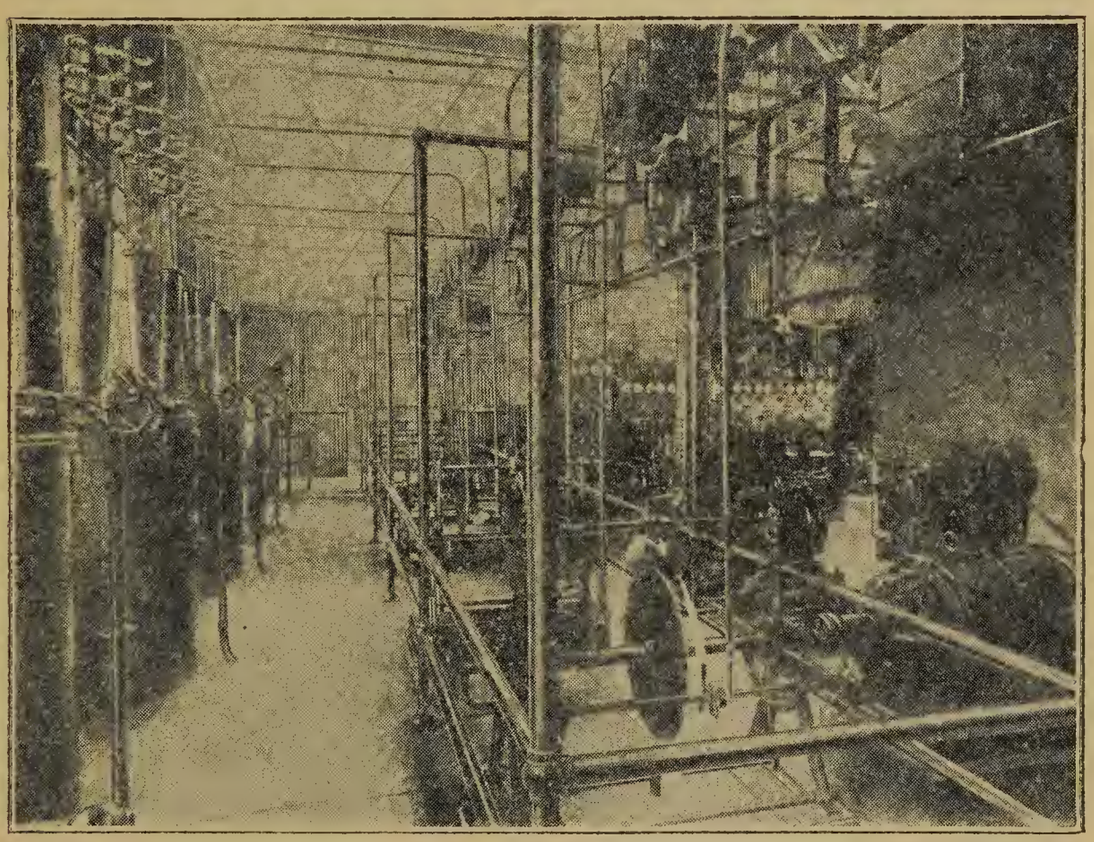


FIG. 15. General view of the electrical precipitating apparatus at the plant of the International Smelting Co., Miami, Ariz., showing switchboard in the rear, transformers at the left, and motor-generator-rectifier sets in the foreground at the right.

The transformer high-voltage terminals are usually provided with choke-coils for the protection of the windings and terminals against surges and high-frequency oscillations. A very effective method is to provide one choke-coil at each end of a condenser type terminal, the mid-point of the terminal being grounded.

It has also been found desirable to use condensers shunted across the low-voltage side of the line, for the protection of the generating apparatus, especially where the latter is of small capacity. Two condensers are used in series, their mid-point being grounded.

The usual overload protection, generally in the form of circuit breakers placed in the low-voltage circuit, is of course always provided for protection against sustained overloads or short-circuits in the treater.

The electrical apparatus provided for the dehydration of oils is usually quite simple, as alternating current is supplied at moderate voltages, usually from 10,000 to 15,000 volts, directly to the treating apparatus. Power is frequently taken directly from a commercial circuit at these voltages, where such is available. Auxiliary apparatus would include measuring instruments, means for voltage variation, and short-circuit protection, such as circuit breakers or fuses. Where the emulsion is agitated, a small motor, usually of two or three H.P., with suitable control apparatus, will also be required.

DISCUSSION.

SAUL DUSHMAN[1]: The remarks of Mr. Braley on the applicability of the kenotron to precipitation are not accurate.

The kenotron differs from the mercury vapor rectifier in two essential respects:

Firstly, the magnitude of the current that may be rectified varies with the temperature of the filament, as shown in Fig. 1. Hence at constant filament temperature there is a definite upper limit to the amount of current that can pass through a resistance load in series with the kenotron.

[1] Res. Laboratory, General Electric Co., Schenectady, N. Y.

Secondly, the voltage drop across the kenotron when operating in series with a resistance on a transformer increases according to a definite law, with the amount of current taken by the load.

These characteristics make it possible to operate two or more kenotrons in parallel and thus increase the amount of current that can be rectified. As well known, it is impossible to operate

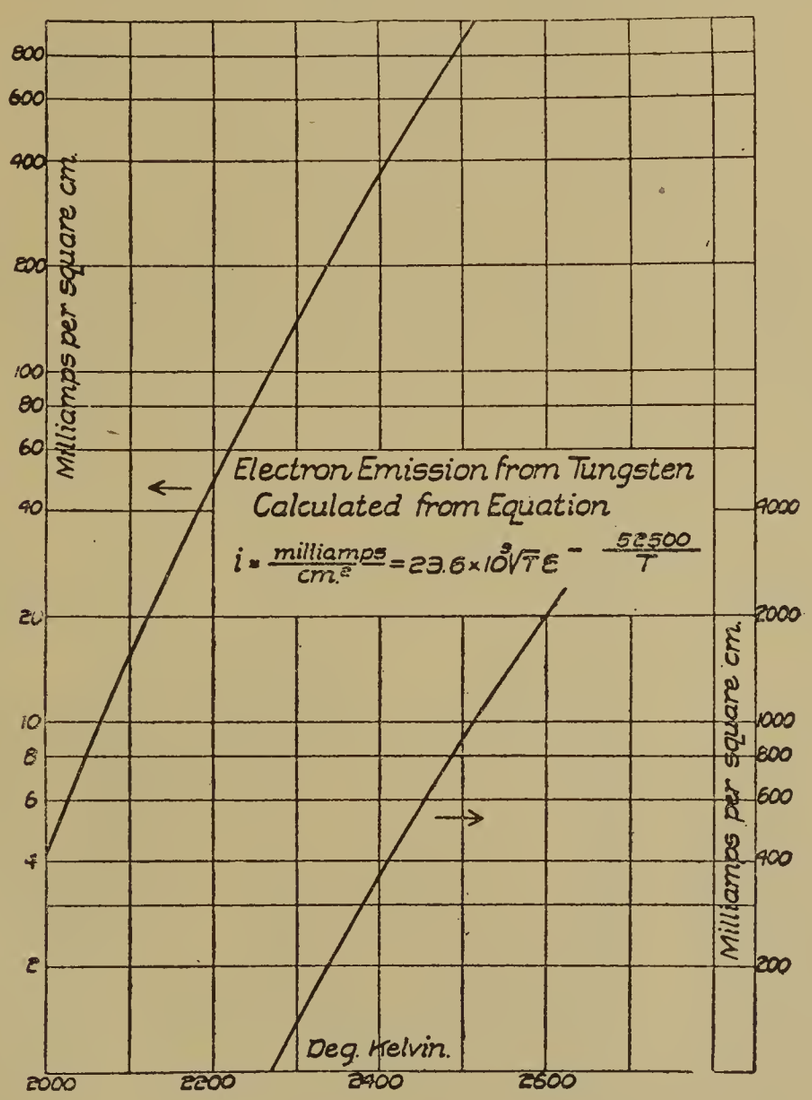


FIG. 1. Electron Emission from Tungsten in a "Perfect" Vacuum.

mercury arc rectifiers in parallel. Furthermore, while the latter type of rectifier is not able to operate on voltages in excess of 10,000, the kenotron has been operated at as much as 150,000, and there is theoretically no upper limit to the voltage at which it can be operated.

Now with regard to the other "serious limitations" mentioned by Mr. Braley.

1. Fracture of the bulbs by very high voltage static discharges does not occur in a well-exhausted tube and is a very rare occurrence in actual operation.

2. The temperature at which the filaments are normally operated is 2400°K. while the melting-point of tungsten is around 3600°. The normal temperature of filament operation is in fact

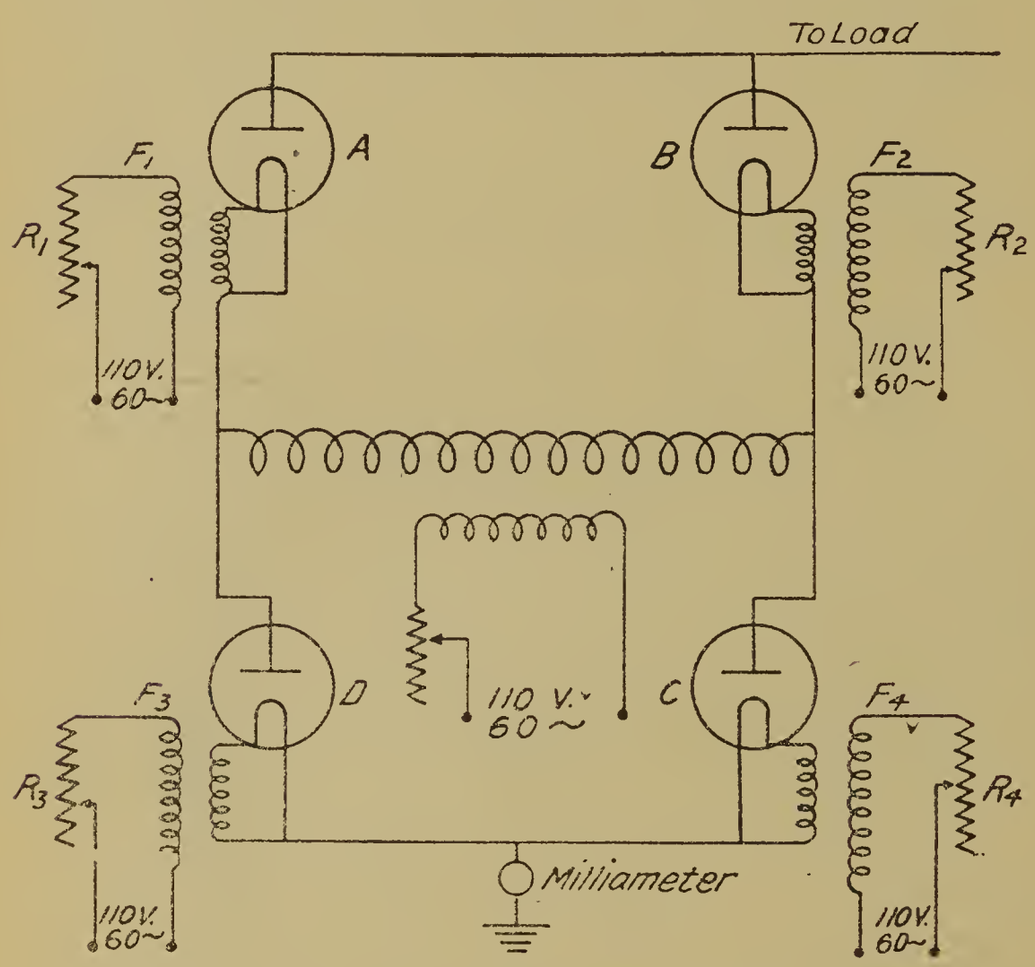


FIG. 3. Arrangement of Four Kenotrons and Transformer for Full Wave Rectification.
F_1, F_2—High tension filament transformers.
F_3, F_4—Low tension filament transformers.
R_1, R_2, R_3, R_4—Resistances for controlling filament current.

about the same as that at which standard incandescent lamps operate with an efficiency of 1 w. p. c. Owing, however, to the fact that much larger diameter filaments are used in the kenotron than in incandescent lamps, a much longer life is to be expected and service tests have substantiated this expectation. Furthermore, such voltage regulation as is used on ordinary lighting circuits is sufficiently satisfactory for operating the filaments of kenotrons.

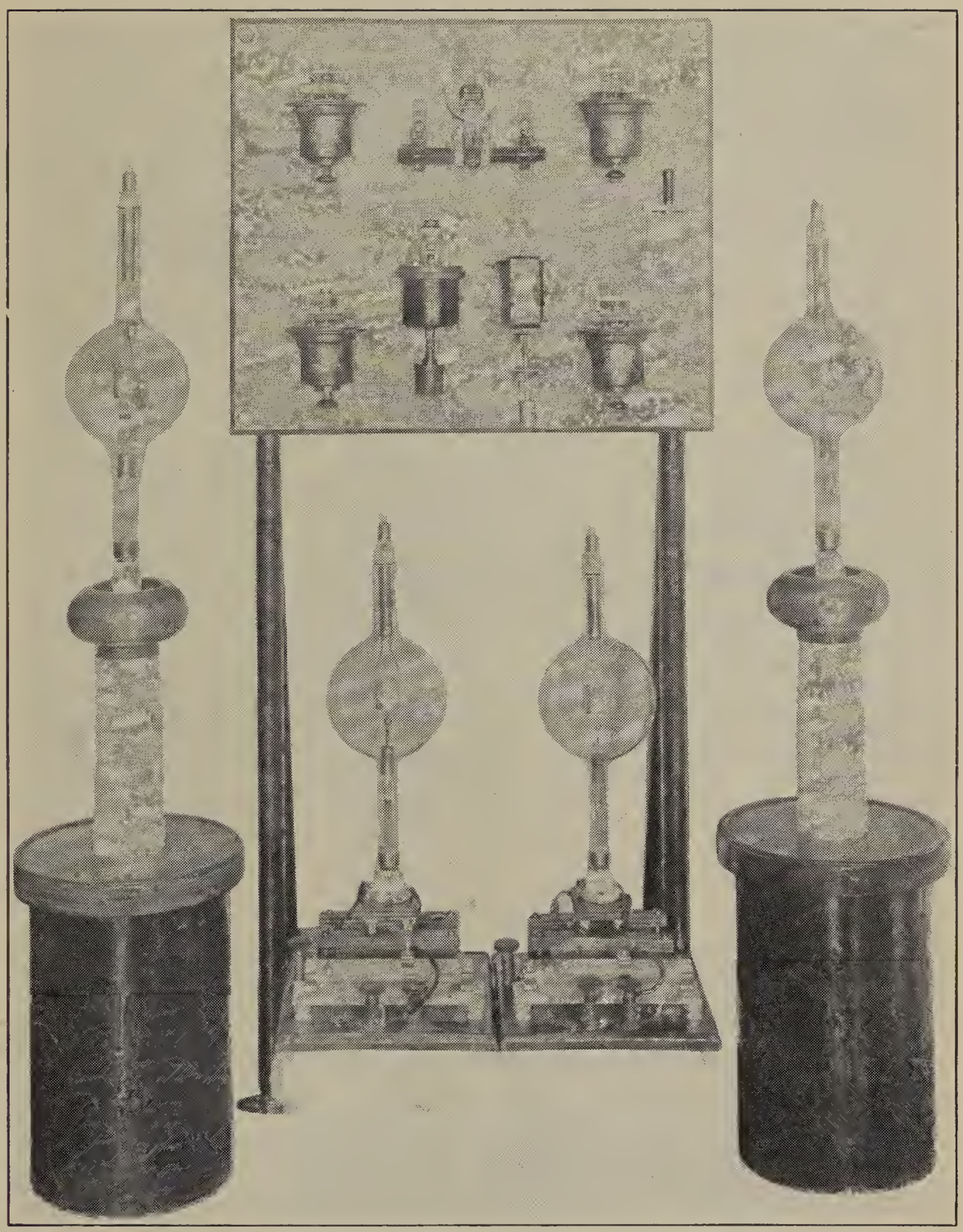


Fig. 2. Switchboard Panel with Protective Devices for Use on Kenotron Precipitation Outfits, with Filament Transformers for Kenotrons.

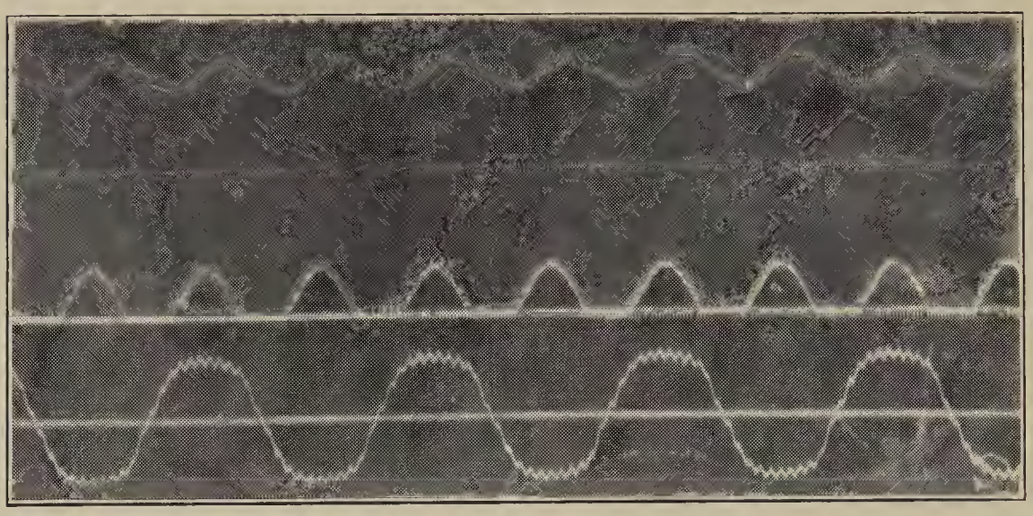


Fig. 4. Four-Kenotron Operation at 50,000 V.

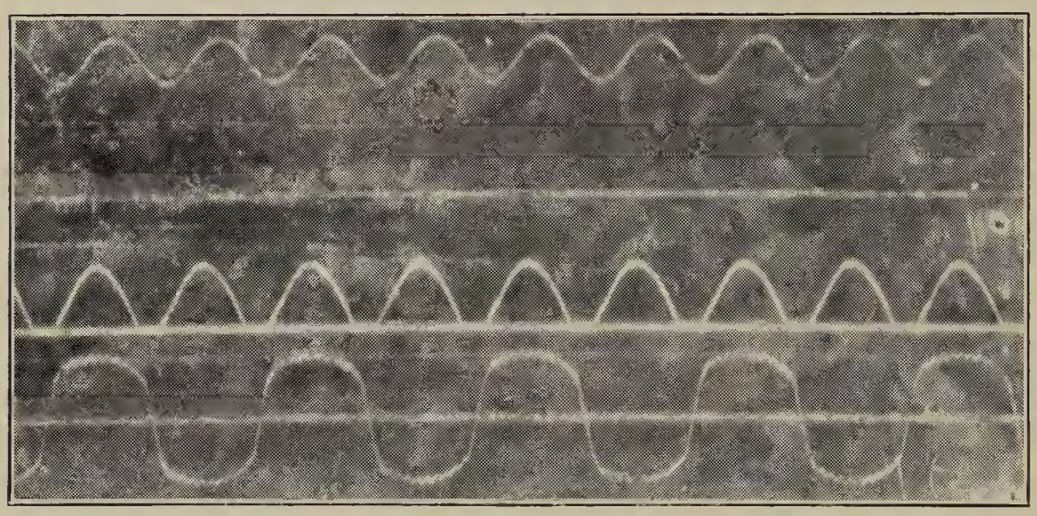


Fig. 5. Four-Kenotron Operation at 100,000 V.

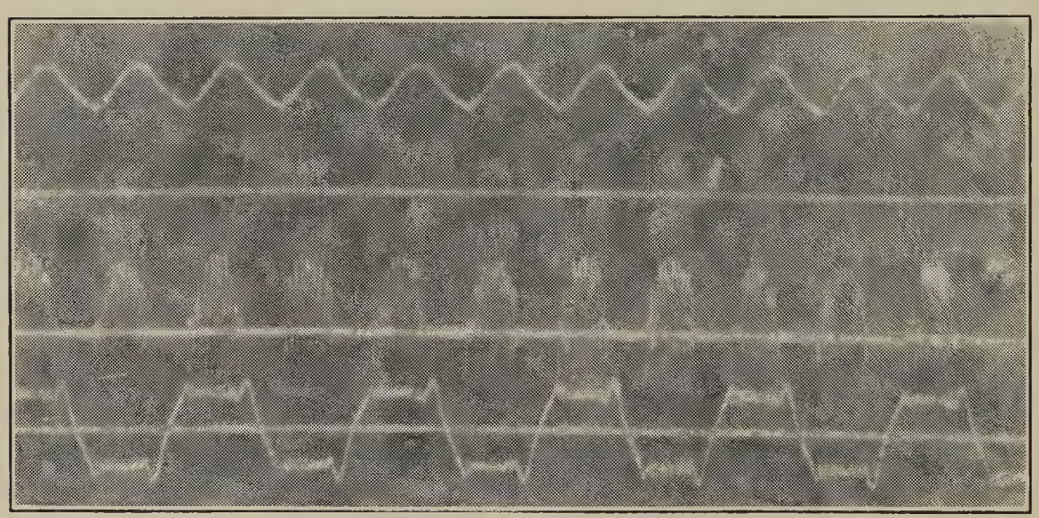


Fig. 6. Mechanical Rectifier Operation.

3. The apparatus for protecting the kenotron equipment against short-circuits in the treater is simple. A relay switchboard developed for this purpose in the Research Laboratory of the General Electric Co. is shown in Fig. 2. This switchboard also provides against burning out of filaments and too low voltage in the filament lighting circuit.

4. This slide also shows an assembly of four standard tubes which are required for an average treater outfit. These four tubes are connected to the high tension side of the transformer in a manner similar to that used with aluminum rectifiers, as shown in Fig. 3. With this arrangement it is possible to rectify up to 0.2 amp. at 100,000 v. or 20 kw. While this unit has been adopted as a tentative standard it is feasible in the case of lower voltage equipments to design kenotrons which shall rectify much larger currents, thus avoiding the necessity of increasing the number of units.

A couple of years ago the writer and some of the engineers of the Research Corporation took a number of oscillograms in connection with experiments on smoke precipitation at a round house. Figures 4 and 5 illustrate the character of the operation of the kenotron equipment as compared with that of the mechanical rectifier, Fig. 6. The lower curves show the primary voltage, the center curves show the current through the treater and the upper curves show the voltage on the treater. The complete absence of surges with the kenotron equipment is very striking, and it is self-evident that with the kenotron equipment failure of transformers due to surges and high-frequency oscillations are reduced to a minimum.

E. E. Thum[2] (*Communicated*): Mr. H. D. Braley's paper is a valuable contribution to the literature on electrostatic precipitation in that it gives a very good general view of the whole subject. Of course it must be recognized that detailed studies of a great many different aspects of a precipitation problem must be studied individually and in detail.

Such widely divergent practice as gas velocities ranging from 2.5 to 10 feet per second and electrical capacity varying from 175 to 600 voltamperes per 1000 cu. ft. may be in part due to the

[2] Associate Editor, Chem. and Met. Engineering.

fact that our ideas as to correct precipitation practice are not yet definitely fixed. In many cases this is doubtless because of a lack of precise information as to the actual mechanism of electrical precipitation. A further instance in point is the diverse preferences for electrical systems of the Research Corporation and of the International Precipitation Company. Of these two organizations each engaged exclusively in the exploitation of Cottrell's process, the former prefers a rectifier driven by small synchronous motor, while the latter prefers the motor-generator-rectifier set. It is hard to properly appraise the standpoint of the protagonists of the various electrical systems, since it is not at all clear exactly what electrical characteristics will give the highest efficiency in precipitation. If the wave-form has a determining effect, commercial current whose characteristics may be suddenly distorted by outside loads should be rectified. However, this current can be used to drive single-phase generators especially designed to produce a correct wave form. Thus it may be that a small motor-generator may produce a poorer wave form and more oscillations building up dangerous electromotive forces than a large alternating current generator producing enough energy for the whole plant, and thus show to a poorer advantage. On the other hand in units of large capacity, operating troubles in one section may modify the current delivered to all the others from a central generator, in which case individual motor-generator sets would become more satisfactory.

The point of view has been strongly held that it is more important to maintain the applied voltage at the treater close to the breakdown point than it is to produce a mathematically correct sine curve. If this be true it is readily seen that the best electrical equipment will be one which damps out the surges and decreases the loss of voltage across the rectifier.

One is inclined to take exception to Mr. Braley's optimistic statements regarding the profits in electrical precipitation. His figures covering the installation at Miami are those of a most modern plant operating on non-corrosive dust. Were their problem to handle a very corrosive fume the story would be entirely different. Even as it is, it is difficult to understand the propriety of crediting the electrical installation with 12 cents per pound for the recovered copper, since this material must be smelted,

shipped, refined, and sold on a market which might normally be taken as 14 cents. Again, simply because the Anaconda Copper Mining Company is able to produce enormous quantities of arsenic, this does not signify that their profits will be correspondingly enormous, for it is notorious that large offerings of little-used commodities will depress the market value to the point where profit vanishes. In this respect a parallel might be drawn from the experience which Western smelters have had with their bag houses. While bag houses have been extremely valuable in collecting dangerous fumes, and are indispensable in the manufacture of pigment by zinc burning, still there are few who would contend that a bag house should be added to a modern non-ferrous smelter with the expectation of deriving a monetary profit therefrom. While I do not wish to be interpreted as saying that an electrical precipitator will be a source of expense rather than of income in every case, I do want to point out that the conditions of operation vary extraordinarily. The cost of operation is usually underestimated, while the value of the recovered product can be easily juggled by an accountant.

An instructive case in point was cited by Mr. B. L. Sackett at the 1918 Colorado meeting of the American Institute of Mining Engineers, where he gave figures for the operation of two good-sized installations. In a period of operation of a little less than two years a sintering plant treater returned profits of approximately 38 percent per annum of the total first cost of the treater, and at a converter treater they have been about 56 percent of the first cost per annum. The figures given represent the value of the metals contained in the recovered fume less the treater operating costs, including repairs, the cost of smelting the fume and making proper allowance for the metallurgical losses in the smelting operation. The repair cost in one case was very high, equalling 40 percent of the first cost of the equipment in less than two years' operation. These returns are figured on war prices for the recovered metal. Had the average metal prices for the last ten years been used one treater would have shown a slight loss while the profit on the other would have been about 8 percent. These figures do not include any value for depreciation of the treater itself, which when handling corrosive gases could easily amount to 25 percent per year.

In the writer's opinion the Cottrell process has permanently established itself as a mitigator of nuisances. Its value in the recovery of by-products is becoming recognized. Its larger future application will probably be as an integral link in a productive chain. In the recovery of by-products electrical precipitation undoubtedly will ultimately yield very large quantities of potash from cement kilns at a price which can advantageously compete with imported salts. As an integral unit in a metallurgical operation one may confidently look forward to its use in the collection of metallic fume produced in the treatment of low-grade and complex ores by processes which have long been known but which could not be used owing to the lack of adequate methods of collecting the volatilized metals.

H. D. Braley (*Communicated*): In regard to Mr. Dushman's remarks concerning the kenotron rectifier. The parallel between the mercury vapor rectifier and the kenotron was stated merely to show the fundamental similarity which exists, *viz.*, that they both readily admit of the passage of current in one direction only. This comparison was cited simply to illustrate this fundamental principle of the comparatively recently developed apparatus with that of the mercury arc rectifier which is well known. It was not intended to convey the impression, as Mr. Dushman's remarks seem to indicate, that the limitations of the mercury arc rectifier apply to the kenotron rectifier. The writer, however, desires to take exception to the statement that it is impossible to operate mercury arc rectifiers in parallel, inasmuch as this is entirely feasible where the transforming apparatus has the proper characteristics for affording parallel operation, and, in fact, parallel operation is being successfully conducted to a large extent in commercial practice at the present time.

The applicability of the kenotron for high-voltage rectification is unquestioned, but it should be borne in mind that the rectification of current for a resistance load is entirely a different proposition than that encountered in most commercial precipitation installations wherein the treater conditions are continuously varying. Arcing and flashing in the treaters, due to causes entirely independent of the rectifying equipment, is often somewhat of a continuous performance. Smoke precipitation is one of the easiest

problems encountered in the practical application of the precipitation process and it is not to be inferred that a device which will successfully precipitate smoke will be equally successful in the precipitation of cement dust and metallic dust from fume, inasmuch as the latter offers far more difficult problems than the precipitation of smoke, and such installations furthermore demand a minimum of service interruptions even of short duration.

Referring to comments of Mr. Thum in regard to the profits to be obtained from precipitation process the writer agrees that caution should be used in estimating such profits. No general conclusion can be drawn which will apply to all cases, for each must be considered on its own merits and with due regard to local conditions which prevail in any particular instance. Figures giving profits or deficits are not easy to obtain, as the tendency to suppress the publication of such information exists almost equally between those installations which pay and those which do not, and for very obvious reasons. The figures cited are those which have already been published and they have been given practically verbatim.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

THE NELSON ELECTROLYTIC CHLORIN CELL.[1]

By C. F. Carrier, Jr.[2]

Abstract.

A detailed description of the history, development, construction and operation of the Nelson cell, including particularly the life of its different parts, the ampere efficiency and the energy efficiency with which it works. The small floor space per unit of output is stressed, also the reliability and dependability of its performance.—[J. W. R.]

Introduction.

Such a number and variety of electrolytic chlorin cells have been described in patent and technical literature, or presented before scientific societies from time to time, that it is with some diffidence we ask attention to this somewhat threadbare topic.

Most chlorin cells have been introduced to the scientific world in their early patent or experimental stages, and many remarkable devices have been described and efficiencies claimed which for some reason have seldom materialized in practice. The Nelson cell is not a novelty of the hour, nor does it yield any theoretical results which have not been previously claimed for other cells, but surely, as the most extensively used chlorin cell in the world, it must possess some features of practical importance that are worthy of careful analysis, and we therefore give publicly for the first time a complete history, description and record of its commercial application. It has been the author's privilege to erect and operate three large plants using this cell, therefore the subject has been considered from intimate practical knowledge, as well as with the co-operation of the inventor.

[1] Manuscript received February 28, 1919.

[2] Technologist, American Cyanamid Company.

History.

This cell was developed by Mr. Harry R. Nelson of the Warner Chemical Co., at Carteret, N. J. It was the desire of this company to secure a supply of comparatively pure chlorin gas for use in the manufacture of chlorin chemicals other than bleach. With this aim in view, they began the production of chlorin about 1905 and it might be mentioned in passing that this, one of the oldest chlorin plants in the United States, has been successfully operating for fourteen years in New York harbor with power that was not hydraulically generated.

The first installation consisted of McDonald cells.[3] These are essentially three compartment cells. Two perforated metal cathode plates support the asbestos diaphragm and divide the cell into a middle anode compartment and two outer, or cathode compartments. Graphite anode blocks with lead lugs were suspended from an earthenware cover box and the diaphragm was held in place by masses of cement concrete in the bottom and ends of the anode compartment. The cathode compartments were filled with caustic soda solution to about the same level as the brine in the anode compartment. Besides a very low ampere efficiency, this cell gave many operating troubles and the up-keep was excessive.

In 1908 it was determined to investigate the relative merits of mercury cathode and diaphragm cells. Mr. Nelson worked on the improvement of the diaphragm cell and the author developed a mercury cathode cell. As a result of these tests, it was decided by all concerned that the diaphragm type of cell was better adapted to meet the needs of the Warner Chemical Co. and further effort was directed to the improvement of the diaphragm cell. The McDonald cells were so materially altered that they completely lost their identity, and a new cell was built which was the progenitor of the modern Nelson cell.

Much of the trouble with the McDonald cell was attributed to the use of concrete in the anode compartment. This was eliminated in the Nelson cell of 1908 by making the perforated metal cathode plate U-shaped and closing the ends of the anode compartment thus formed by means of slate slabs. This was a great

[3] U. S. P. No. 697,157 of Apr. 8, 1902.

improvement, but the caustic compartment was still filled with caustic liquor and the ampere efficiency was still lower than was desired.

It was soon found that better results were obtained by leaving the caustic compartment empty and allowing the caustic and brine solution to drip from the diaphragm, but there was still some trouble due to decreasing percolation and increasing resistance as the operating period lengthened.

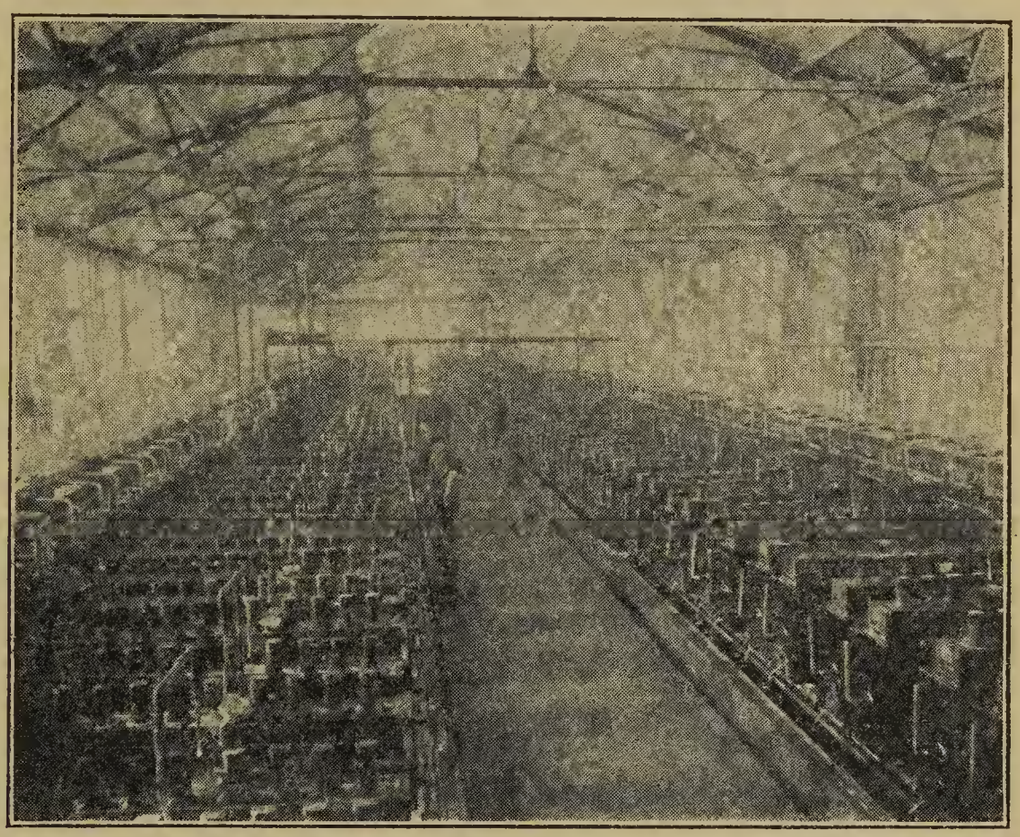


FIG. 1. Installation of Chlorin Plant.

In 1913 these difficulties were overcome by the introduction of steam into the cathode chamber and the cell assumed the final form to be described below. This unique improvement was followed by the discovery of a process for treating the graphite anodes which more than doubled the life of the graphite, and later by the development of an automatic feeding device, details of which will be discussed further below under "description" and "special features."

The culminating development has been during the period of the war, there now being in operation over twenty plants in var-

ious parts of the world. Nearly 8,000 cells have been installed, rated at 1,000 amperes capacity and representing a daily output of about 200 tons of chlorin gas. The largest single chlorin plant in the world is the United States Government plant at Edgewood Arsenal, Baltimore, Md., where over 3,500 Nelson cells were installed for the production of 100 tons of chlorin gas per 24 hours. The construction of this plant was one of America's greatest chemical achievements during the war. It was built at a moderate cost, only 7 percent above the estimates, which had not included overtime; it was ready to operate on the date promised and was completed without alteration of the plans after construction had been started. The other large chlorin plants of the world have been built in sections from time to time over periods of many months while the Edgewood plant, the largest of them all, was entirely built in four brief months.

Description.

A description of the cell is to be found in U. S. P. No. 1,149,210, of Aug. 10, 1915, but a much better idea of the general construction can be obtained from the accompanying illustration. Special attention is called to the simplicity of the design. A substantial, rectangular tank made of ¼-inch (6 mm.) steel plate forms the body of the cell. In this is mounted a U-shaped cathode plate of perforated sheet steel, which also acts as the form for the anode compartment. The asbestos diaphragm is supported by the cathode plate and the ends of the U-shaped anode compartment are closed by blocks of cement mortar. An inverted, rectangular box formed of slate slabs closes the top of the anode compartment and also acts as a support for the graphite blocks which form the anode. The anode structure is so clearly shown in the illustration that it only remains to be said that the blocks are 4 x 4 x 17 inches (10 x 10 x 43 cm.), the lugs are 2½ inches (6 cm.) diam. by 12 inches (30 cm.) long, and the immersion is about 15 inches (38 cm.). It is to be noted that there is but one joint in the graphite, thus insuring maximum conductivity. The positive terminal is connected by means of a flat copper bar bolted to the lugs of the anode blocks, while the copper plate riveted to the perforated cathode performs a similar service for the negative terminal.

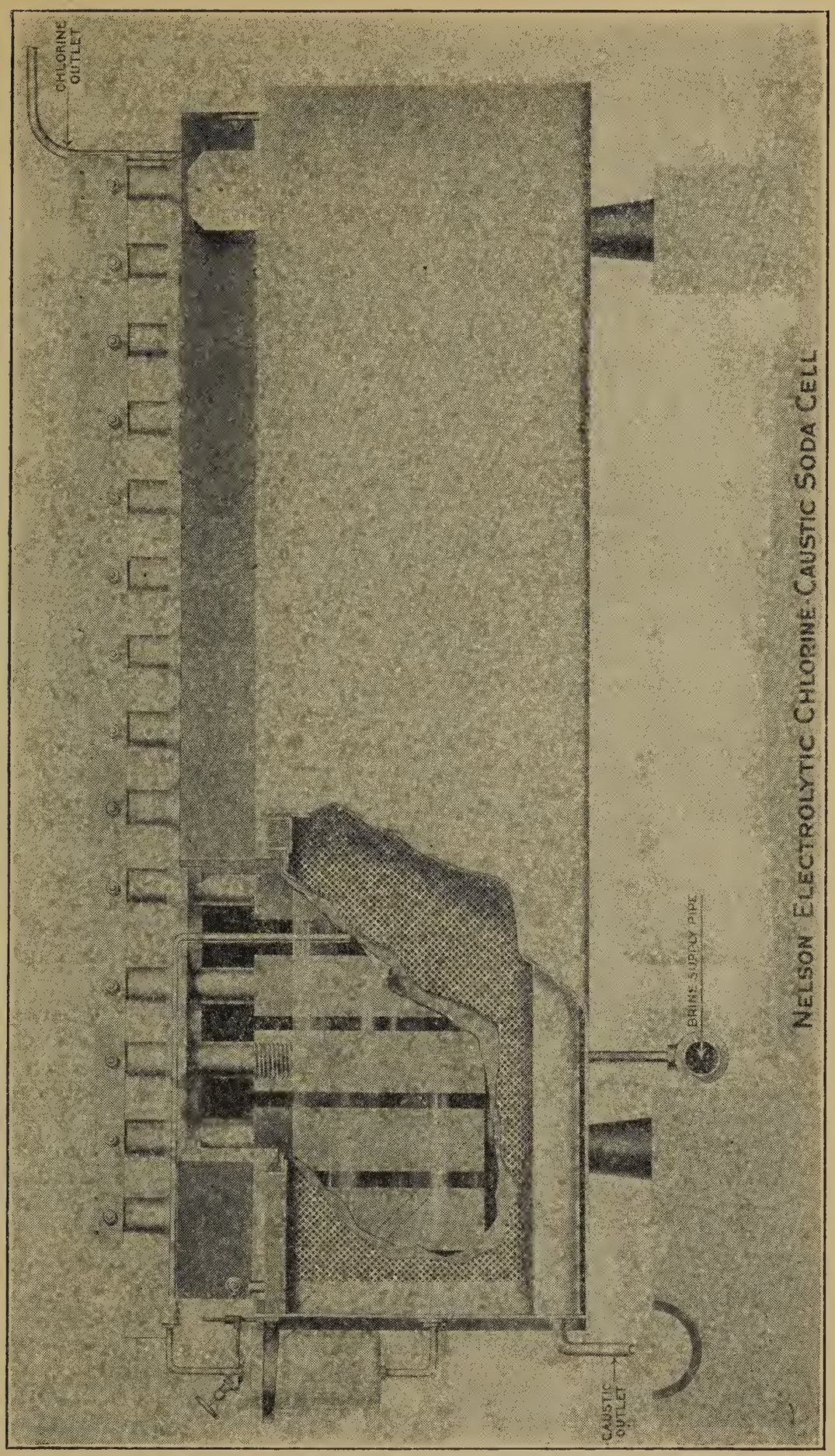

CHLORINE OUTLET
BRINE SUPPLY PIPE
CAUSTIC OUTLET
NELSON ELECTROLYTIC CHLORINE-CAUSTIC SODA CELL

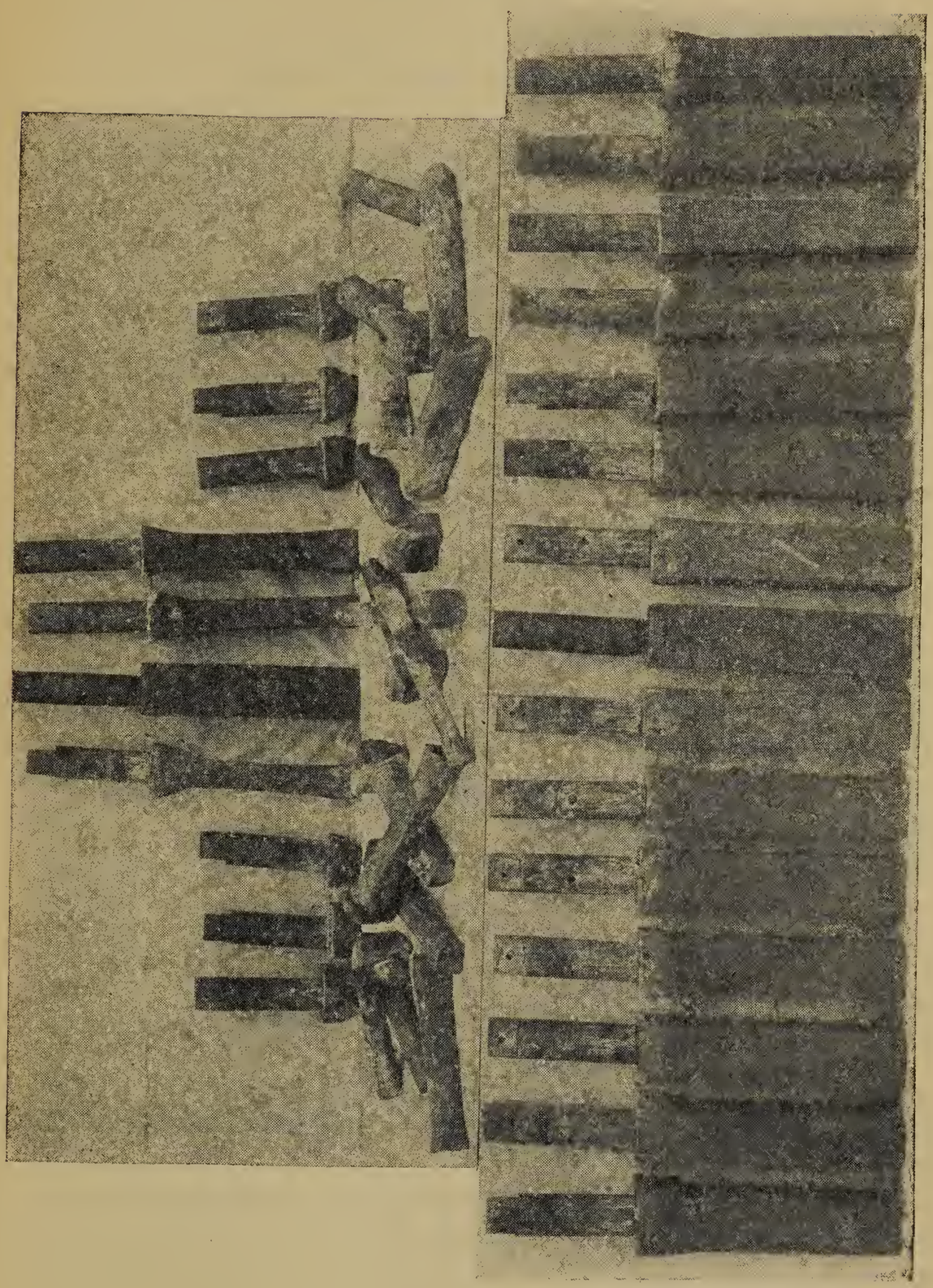
FIG. 2.

The latest built cells, like the illustration, are operated with an automatic feeding device which is very simple and which has proved to be entirely reliable. It consists of a strongly buoyant float on which is mounted a dull knife edge which presses against a bit of rubber tubing through which the brine is supplied to the cell. The slightest rise of the float cuts off the feed entirely and the least drop permits the flow to start again. No trouble is caused by dirt in the brine because all of the brine is purified before passing to the cells, it being quite essential in the operation of any electrolytic cell for chlorin manufacture that iron, alumina, lime and magnesia be substantially absent.

The brine percolates through the diaphragm, and the rate of flow is kept uniform by means of an atmosphere of steam that is maintained in the cathode compartment. This steam also helps to dissolve the caustic soda formed at the cathode as well as to heat the cell and thus reduces the resistance of the cell. In case the pores of the diaphragm begin to close, as will be shown by excessively strong caustic liquor, they can be freed by increasing the amount of steam.

The normal operating period between renewals of the diaphragm is from six to eight months, and during this time the operating conditions remain practically constant. The cells are rated at 1,000 amperes (72 amperes per anode), at which point the most efficient results are obtained, but they can run as low as 600 amperes without serious loss of efficiency, and will stand a continuous overload of 10 percent without damage to plant or product. The voltage drop, averaging the switchboard reading and including all connections and cables, for a series of sixty cells, is from 3.5 to 3.8 volts per cell, and will usually average 3.7 volts over a period of six months. Only careless operation will cause an increase of more than 0.2 volt in the course of a six months' operating period.

The commercial guarantee under which these cells are sold specifies that when operated at 1,000 amperes and supplied with not over 120 lb. (55 kg.) of salt, that each cell will produce 60 lb. (27.3 kg.) of chlorin and 68 lb. (30.9 kg.) of sodium hydroxide per 24 hours. For the first few days the caustic liquor flows very



FIGS. 3 (above) and 4 (below).

freely and is abnormally high in salt, but when the cells have reached normal working conditions the liquor will contain 10 to 12 percent NaOH and 14 to 16 percent NaCl.

Technology and Special Features.

Operating Period and Shut Down: The normal operating period is six to eight months. At the end of this period it is considered better to renew the diaphragms and wash out the cells even if the operating conditions still seem very good, for they may become worse very rapidly if the period is stretched too far. The ordinary anodes are simply impregnated to prevent the electrolyte from creeping to the copper contacts, and only part of these will last more than two operating periods, that is 12 to 15 months. A new process of treating the graphite has been perfected which increases the life of the anode to over two years. (See paragraph on anode, and illustration).

One of the best features of this cell is that it can be shut down either accidentally or intentionally for short or for long periods as often as desired, without damage to the cell or serious loss of efficiency. To the best of our knowledge this is the only commercial chlorin cell that can stand frequent shut downs without serious consequences.

Simplicity of Structure: Few, simple parts usually mean quick installation and low repair costs. The former was well illustrated at the Edgewood Arsenal plant and the latter has been borne out by the cost records of several large plants over long periods of operation. It also keeps down installation costs and operating expense, as it is possible to operate with ordinary labor. There are no moving parts and the entire structure is rugged and durable.

The Anode: The anode is made up of the most economical shape and size of graphite obtainable. Any change in the relative dimensions results in a higher price per pound. The machine work is extremely simple, permitting rapid production with minimum labor. The original appearance of the anode blocks can be seen in the illustration of the cell and in the center of the two illustrations showing anode blocks only. Fig. 3 shows a set of anode blocks with no special treatment, which have been run 12 months without renewal of the diaphragm. Fig. 4 shows a set of

specially treated anodes which were in service six months before the diaphragm was changed and then for a year before the second diaphragm renewal. They are apparently good for another year at least.

Life of the Cell: The life of the tanks is not definitely known, as some over ten years old are still in service and it is estimated that they will last twenty years. The screens last two to five years, except when accidental local electrolytic action occasionally causes a local hole that would cost more to repair than the cost of a new screen. The asbestos diaphragms are put in new every six to eight months when the cells are washed out. The new-process anodes give efficient service for two to three years. The life of the other miscellaneous parts is largely dependent upon the care with which they are handled and operated. It might be mentioned in passing that the average concrete body cell is comparatively short lived, it being only two to three years before the concrete becomes soft and porous, necessitating the complete recasting of the cell. The reinforcing steel in a concrete cell will usually be found to weigh more than the steel in an all-steel cell.

Floor Space: Allowing ample aisles, the production of chlorin per square foot of floor space per 24 hours is about 2½ lb. (12.5 kg. per sq. meter), while the average concrete cell will require two to three times as much floor space for the same production. A high floor-space factor means a saving in installation of piping, bus-bars, etc., per unit of output.

Attendance: With cells provided with the automatic feed controls, one attendant is ample for two hundred. At the plant of the Warner-Klipstein Chemical Co., South Charleston, W. Va., there is only one attendant for 480 cells.

Quality of the Chlorin Gas: As previously mentioned, this cell was specially developed to produce relatively pure chlorin gas for the manufacture of chemicals. How successfully this was accomplished may be judged from the fact that large groups of cells have been run for periods of several weeks with the gas analyzing 99.5 to 99.8 percent Cl, and have been operated continuously producing gas averaging 99 percent pure. Only ordinary precautions are required to keep the gas substantially free from

air, without permitting the escape of chlorin into the atmosphere of the cell room.

Quality of the Caustic: A considerable amount of undecomposed salt of course passes into the caustic liquor from all diaphragm cells, but this is readily removed in the evaporator. Far more serious is the presence of chlorates or hypochlorites. In spite of the high temperature at which these cells operate, chlorates are not found in the caustic and it is very rarely that hypochlorites can be detected. The surest hypochlorite indicator is the life of the tubes in the evaporator, even small amounts of hypochlorite greatly shortening the life of the tubes. In some caustic plants this is a serious problem, but with this cell a life of two years is not uncommon.

Efficiency: As already noted, the guarantee under which the cells are sold is that when operated as instructed each cell will require 1,000 amperes and not over 120 lb. (55 kg.) of salt to produce 60 lb. (27.3 kg.) of Cl and 68 lb. (30.9 kg.) of NaOH in 24 hours. As a matter of fact this guarantee has always been exceeded.

1,000 amperes requires theoretically 115 lb. (52.3 kg.) of NaCl and should produce 69.86 lb. (31.75 kg.) Cl and 78.67 lb. (35.75 kg.) NaOH. The guaranteed ampere efficiency based on the chlorin is therefore about 86 percent. Plant records actually show about 90 percent ampere efficiency and 60 percent energy efficiency.

The energy efficiency is the important figure. Several commercial cells operate, or claim to operate, at higher ampere efficiencies than the Nelson cell, but the gain is wholly illusory if the energy efficiency drops to 55 percent at the same time. In one case the energy consumed is 1.53 kw. hours per 1 lb. (3.37 kw. hours per kg.) of Cl and in the other case 1.67 kw. hours (3.67 kw. hours per kg.) are required, even though the ampere efficiency was 5 percent better. This means a saving of nearly five cents per cell per day, or about $4,000 per year in a plant producing 6 to 7 tons chlorin per day, assuming power to cost $0.005 per kw. hour.

The operating data over a long period at one plant gave an average voltage drop of 3.7 volts per cell over a period of six months, varying from 3.5 to 3.8 volts. The high average energy

efficiency is due to the fact that the cells start at a very low voltage, which remains almost constant up to the end of the operating period.

Assuming that 2.3 is the correct factor for the theoretical decomposition voltage of NaCl in a water solution, the energy efficiency at 3.7 volts is 62 percent and at 3.5 volts is over 65 percent.

Conclusion.

It is not claimed that absolute perfection has yet been obtained, but it is believed that this cell has reached a stage where experiment and abstract theory are things of the past. It is now a fully developed "machine" for manufacturing chlorin gas, which a chemical manufacturer can purchase as he would a filter press, a still, an evaporator, or any other piece of standard chemical engineering equipment, and be assured of a fixed regular output of high quality, produced in a moderately costing, efficiently operating plant, which can be operated by ordinary labor with economical operating costs and repairs.

The author wishes to acknowledge the assistance of Mr. Nelson, who supplied some of the data and the illustrations for this paper, and also to thank the Warner Chemical Co. for the release of data and information.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

ELECTROLYTIC SILVER AND GOLD REFINING AT PERTH AMBOY, N. J.

By Geo. G. Griswold.[1]

Abstract.

An illustrated description of refining silver bullion electrolytically by the Moebius process, at the works of the American Smelting & Refining Co., Perth Amboy, N. J. The Wohlwill plant for electrolytically refining the gold bullion, and recovering from it platinum and palladium, is also incidentally described.

The first electrolytic plant for the treatment of doré bullion was put in at the Kansas City Smelting and Refining Co., Argentine, Kansas, in 1884 or 1885 by Dr. Moebius. This worked entirely satisfactorily, and was followed by other installations. The second was at the Perth Amboy plant of the American Smelting and Refining Co. These were of the Moebius vertical type. Dr. Moebius afterward patented a horizontal system with travelling silver belts for cathodes, a small installation of which was made at Perth Amboy but did not prove as satisfactory there as the vertical type. When the parting plant was destroyed by fire, in 1897, it was decided to install the vertical system in the new plant. The decision to adopt this system was due to its greater capacity for a given floor space, smaller tie-up, and the fact that being run in connection with a lead refinery, the anodes could be cheaply made 990 fine or better.

The new tank room contains twenty-four sections of six tanks each, the sections being in series and the tanks in multiple. Its arrangement is shown in Fig. 1, which also shows the superstructure above the tanks carrying the leads, switches, and drums for raising and lowering the silver boxes, etc.

[1] Supt. Smelting and Lead Refinery, Am. Smelting & Refining Co., Maurer, N. J. Manuscript received Sept. 12, 1918.

FIG. 1. Tank-Room.

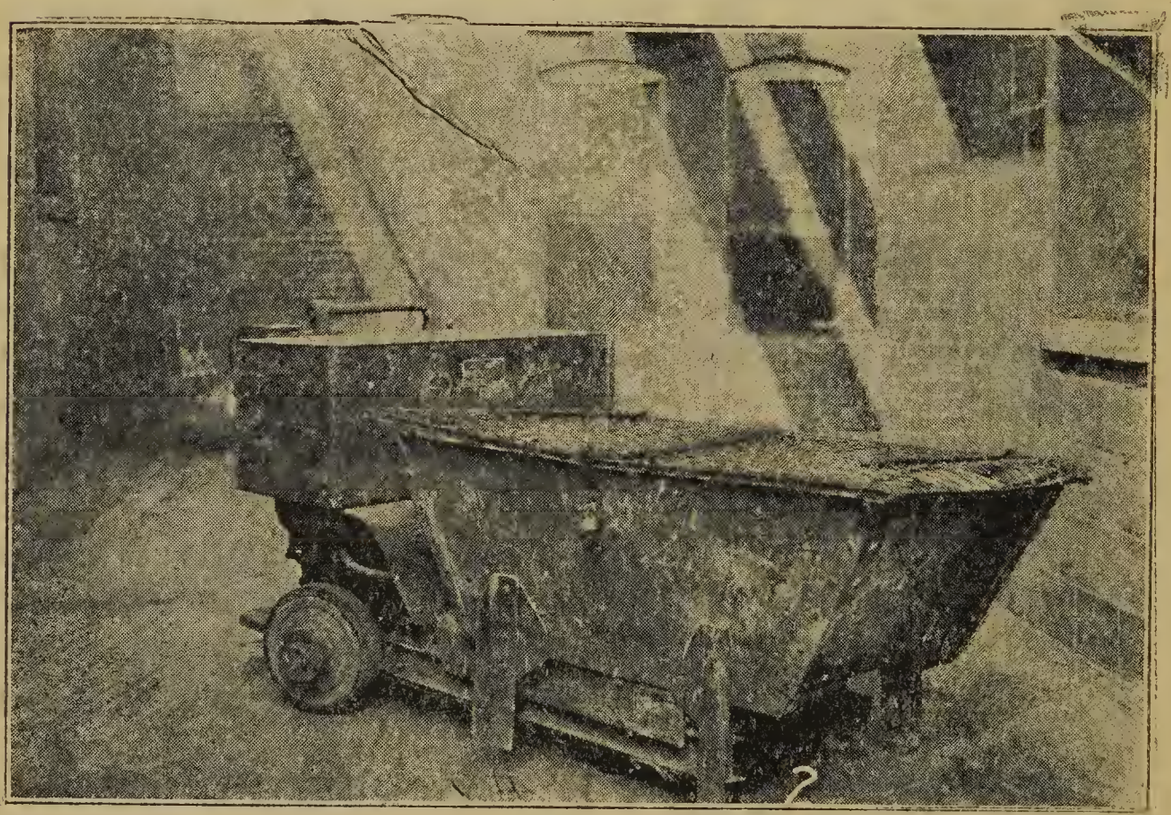


FIG. 2. Truck for Transferring the Anodes.

In the original installation the tanks were made of wood, which was never very satisfactory and consequently stoneware tanks were used in the new plant.

The anodes weighing about 100 ounces (2.8 kg.) are cast by hand in the lead refinery and transferred in locked steel boxes, by a storage battery truck, to the parting plant. They are weighed, counted and punched, and are then used as required. The punching gives a slight depression on each side of the ear of the anode, thus furnishing a secure hold and better contact for the spring clips which support them. The method of transferring the anodes is shown in Fig. 2.

Fig. 3 shows the cathodes, made from cold-rolled silver sheets 1/32 inch (0.8 mm.) thick, bag frames, canvas bags, spring clips which hold the anodes, and the manner in which they are suspended from the cross-contact bars. There are 4 bags and 5 cathodes in each tank, and each bag holds 4 new anodes. Seventy-five percent of the silver is deposited in 24 hours. It is constantly brushed from the cathodes by wooden sticks, rigidly attached to a wooden frame having a reciprocating motion, which serve to prevent short circuiting and also to give circulation. The deposited silver falls into wooden silver boxes, and is removed daily as shown in Fig. 4, by dropping the hinged bottom of the silver boxes, which dumps the silver into a tray from which it is emptied into a wooden silver-car, provided with a false bottom and filter. The car is transferred to vacuum boxes, the silver washed, and then transferred to the melting room, where it is dried, melted, and cast into bars. (Figs. 5 and 6.)

The electrolyte is a neutral nitrate solution containing 15 to 20 grams silver and 30 to 40 grams copper per liter. Silver nitrate is added as required, and enough electrolyte withdrawn daily to keep the purity up to standard. The waste electrolyte is put in a wooden tank and the silver precipitated on copper, after which the copper is cemented out on iron and the resultant solution, after passing over lime, thrown away.

Current was supplied by an engine-driven generator in the same building and a current density of 19½ amp. per sq. ft. (200 A. per sq. m.) carefully maintained; but later, it was found more advantageous to install a motor-generator set in the central power plant, and to increase the current density to 40 A. per

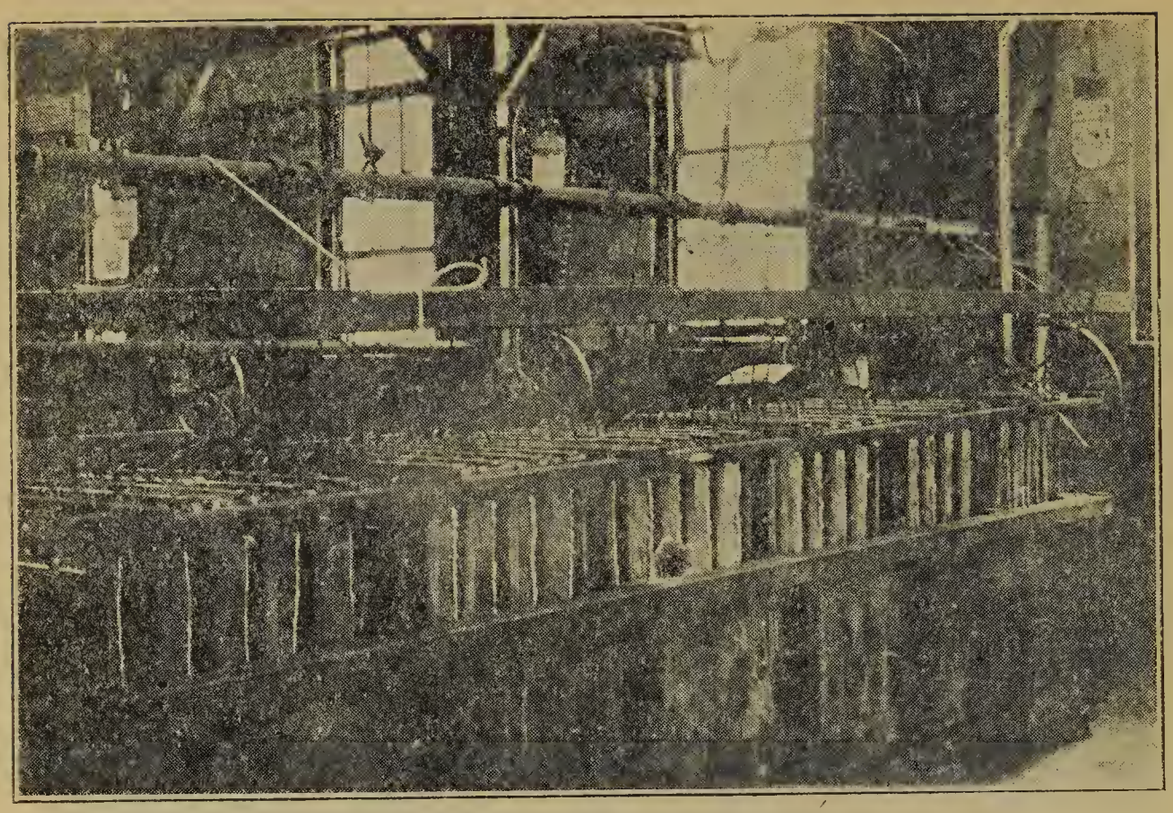


FIG. 3. Showing Suspension of Cathodes.

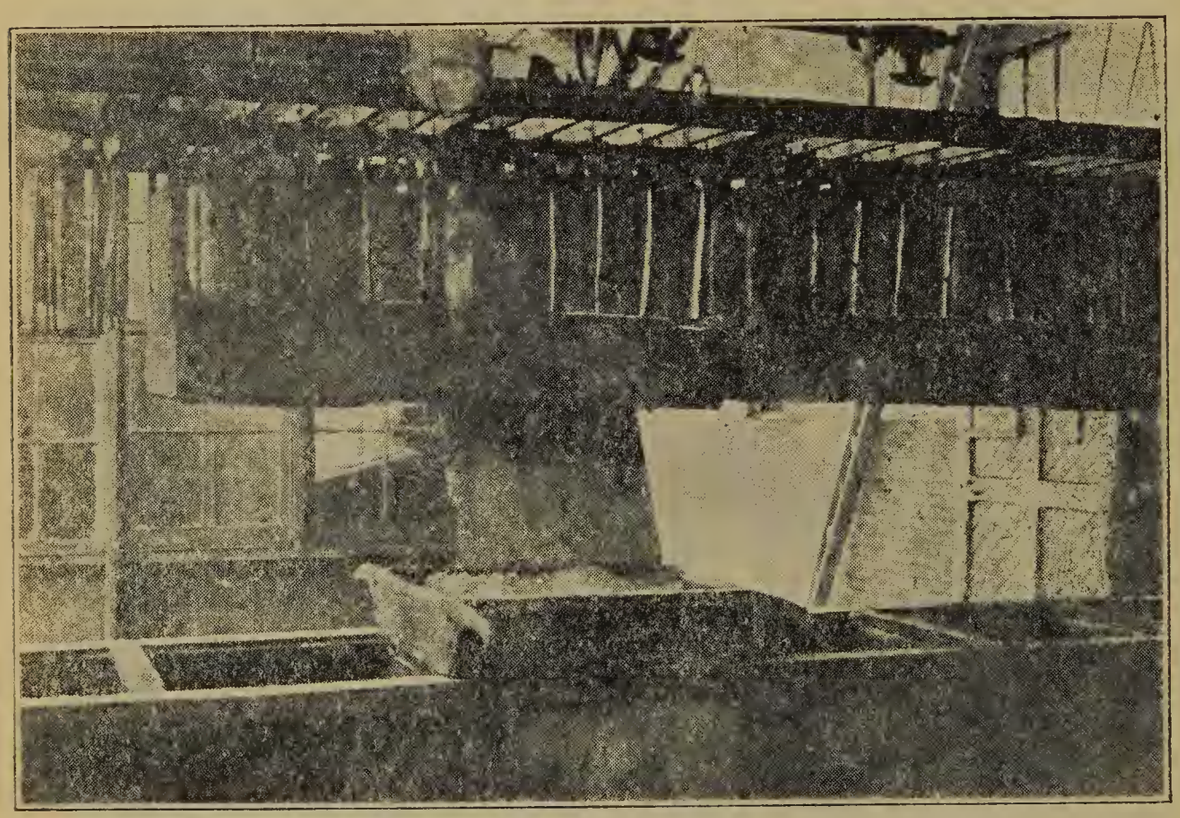


FIG. 4. Method of Removing the Deposited Silver.

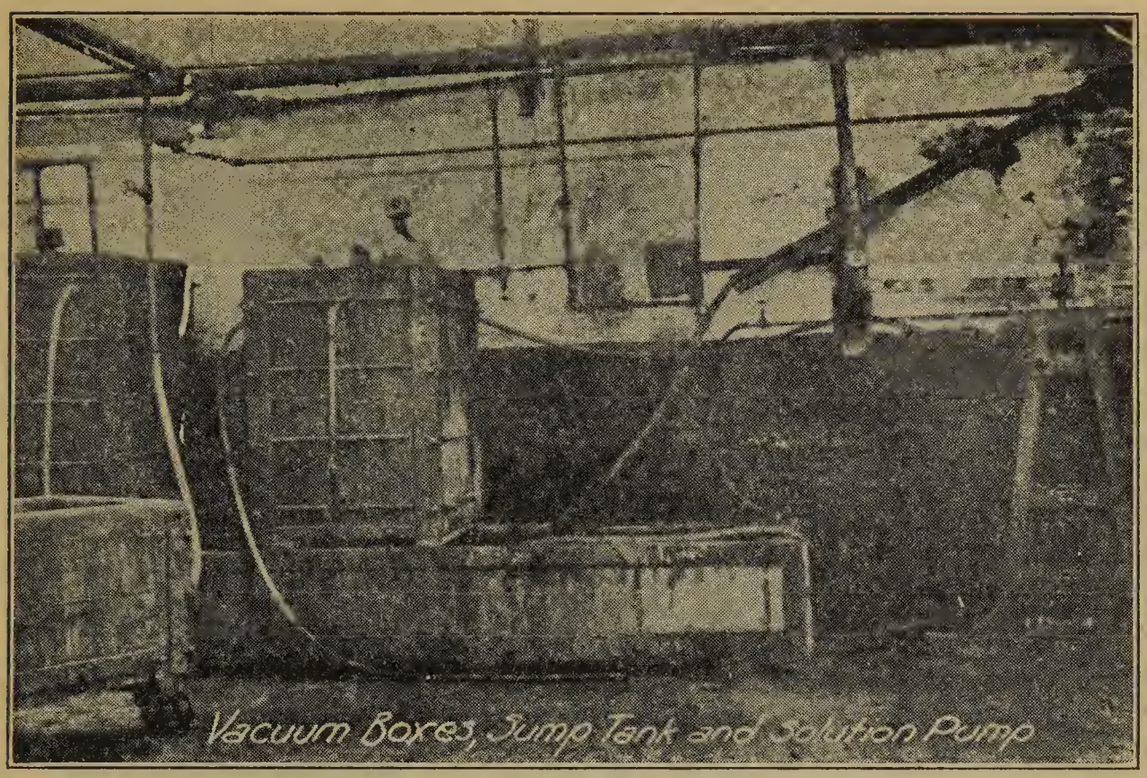


FIG. 5. Vacuum Boxes, Sump Tank and Solution Pump.

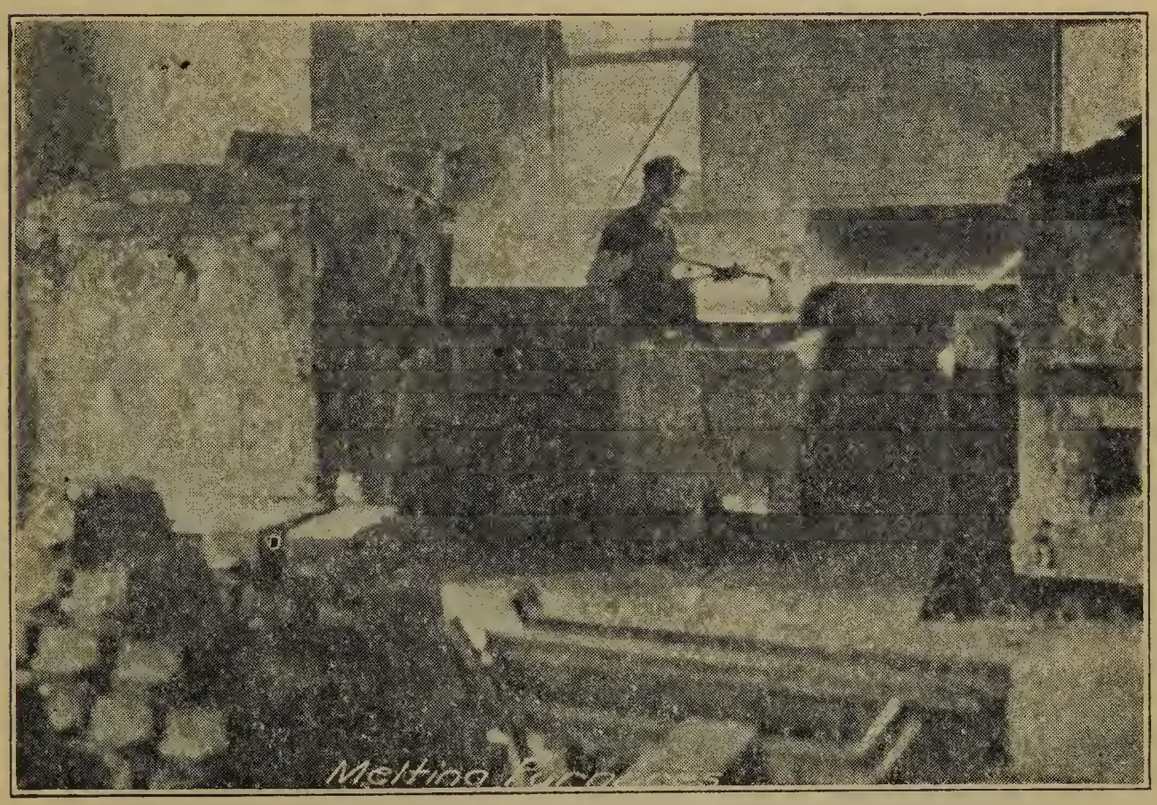


FIG. 6. Melting Furnaces.

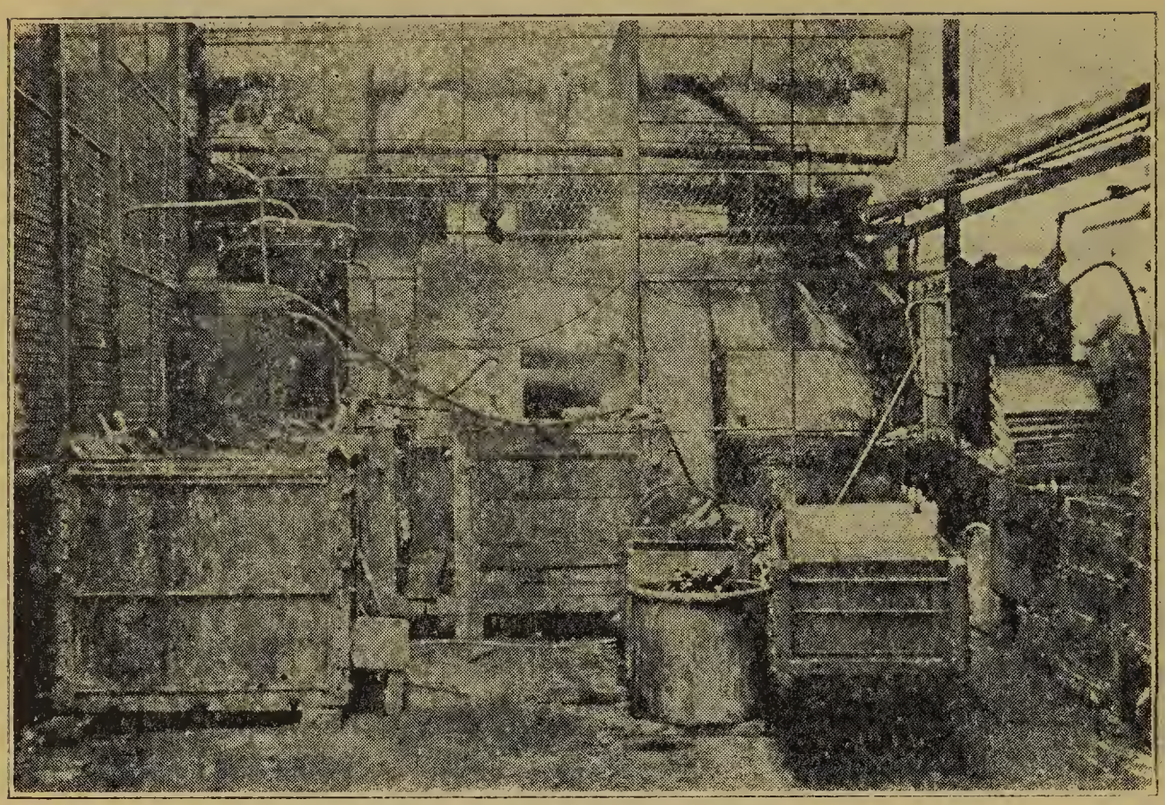


Fig. 7. Gold Box and Boiling Kettles.

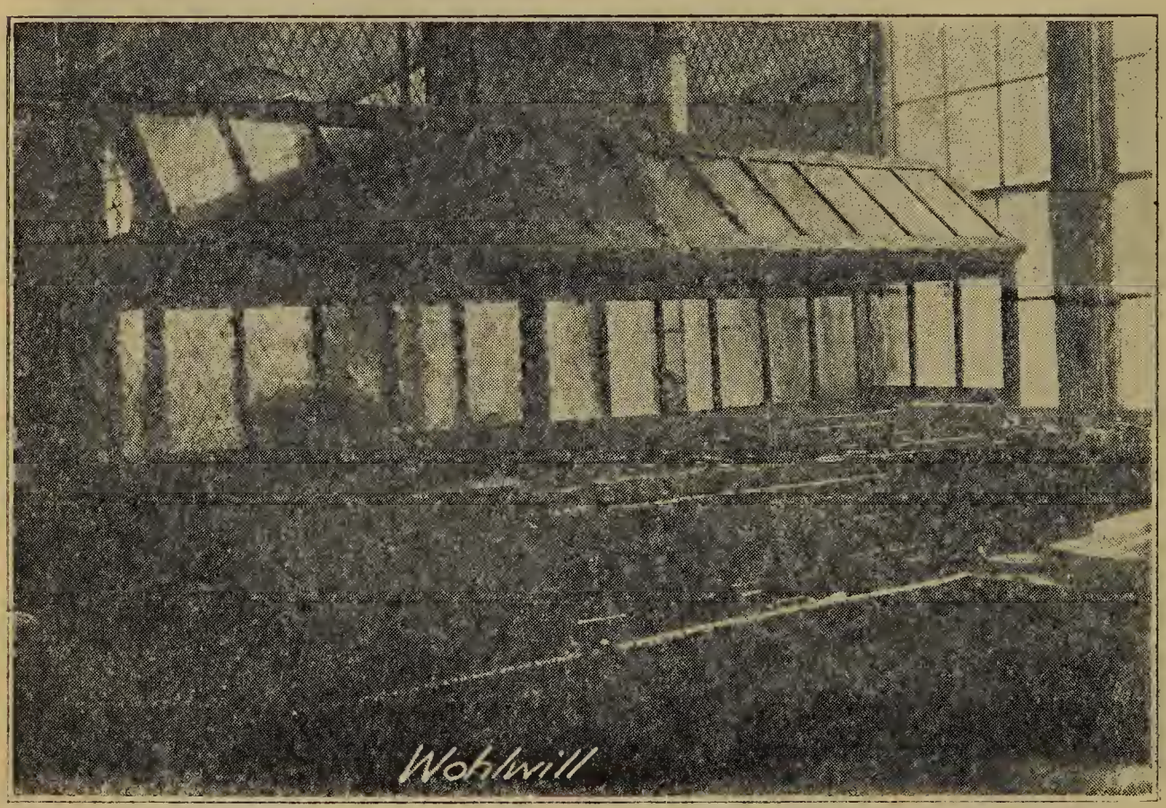


Fig. 8. Wohlwill Gold-Refining Installation.

sq. ft. (440 A. per sq. m.), which increased the capacity accordingly. The current is carried from the switchboard by copper cables which distribute it through mercury-filled copper cups over ⅞ inch (22 mm.) cold-rolled copper bus-bars, the end ones being bent to fit into the mercury cup when lowered into position.

The bags which contain the gold slime are removed at regular intervals, and the contents sluiced into the gold box, thoroughly washed, transferred to iron boiling kettles, and treated with H_2SO_4, 66° B., to remove the copper and silver. (Fig. 7.) The gold slime is washed, dried and cast into anodes for electrolytic refining by the Wohlwill process. The liquor from boiling the slimes is run into lead-lined tanks, and the silver precipitated out on copper. The copper-bearing solution then goes to the copper refinery.

The Wohlwill installation has five cells, and is in series with the silver sections. (Fig. 8.) At the time it was put in, owing to the war, it was not possible to obtain porcelain, and stoneware cells were used. With the high current density of 150 A. per sq. ft. (1650 A. per sq. m.), osmosis develops in the stoneware; also, as they crack with any unequal strain, it was necessary to place them in a lead-lined box filled with water heated by steam coils. The boxes are supported in a lead-covered table with cupped top.

An interesting feature is the use of mercury cups on the ends of the copper bus-bars, whereby any unit or units can be cut in or out quickly by using cross bars, also of copper, with ends bent to fit into the cups.

The cathodes are thin gold sheets, rolled from electrolytic gold, and are connected to the contact bars by bending one end of the sheet around them and clipping them fast.

The electrolyte contains 30 percent free HCl, 80 to 85 grams gold per liter, and varying amounts of platinum and palladium. When the solution accumulates a sufficient percentage of these latter metals, a portion is withdrawn, and they are separated, refined, and sold as pure metals.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

A PROCESS FOR ELECTROLYTICALLY REFINING NICKEL.[1]

By Geo. A. Guess.[2]

Abstract.

Impure nickel, containing copper and iron, is electrolytically refined, using it as anode. Iron goes into solution and accumulates there, but copper, which also goes into solution, is precipitated therefrom continuously by keeping powdered calcium carbonate (limestone) suspended in the electrolyte. An unusual salt is thus formed from the copper in solution, $2CuO.2NiO.SO_3$, completely precipitating out copper, which settles, with $CaSO_4$, as a mud. The cathode is enclosed in a canvas bag to keep it free from mechanical impurities. Glue is used in the solution. The cathode nickel contains usually less than 0.001 percent copper. The mud is reduced by fusion to a nickel-copper matte, which is dead roasted, and then reduced to nickel-copper alloy, NiCu, which is used as anode in an ordinary copper refining bath to recover its nickel as sulphate. [J. W. R.]

If an anode of crude nickel, which may contain copper and iron, is electrolyzed in a bath of nickel sulphate in which is suspended finely divided calcium carbonate, there is deposited on the cathode, if suspended in a sack diaphragm, metallic nickel practically free from copper and iron. This is the basis of a process developed in the Metallurgical Laboratories of the University of Toronto. The action of the finely pulverized limestone appears to be represented by the following equation:

$$2\ CuSO_4 + 2\ NiSO_4 + 3\ CaCO_3 = 2\ CuO.\ 2\ NiO.SO_3 + 3\ CaSO_4 + 3\ CO_2$$

[1] Patented in the United States and Canada, British and Foreign patents applied for.

[2] Professor of Metallurgy, Toronto University, Toronto, Canada. Manuscript received Nov. 25, 1918.

The copper is precipitated as a double basic sulphate of copper and nickel; which is quite insoluble. The function of the diaphragm is to protect the cathode from mechanical pollution by contact with the insoluble copper salt.

Since an equivalent amount of nickel is combined with the copper in the insoluble basic sulphate, it is obvious that the copper content of the anodes should be as low as possible. Preliminary treatment of the converter matte to remove copper is necessary. Two methods at once suggest themselves for doing this. The first is by leaching the roasted matte with dilute sulphuric acid, as is done in the preliminary operation for the Mond process. The second method would be a fusion with coke and salt cake (Na_2SO_4), which is the preliminary step in the Orford process. The data herewith submitted were obtained in refining anodes produced by the first method.

The matte used had the following analysis: Ni, 56 percent; Cu, 24 percent; S, 18 percent. It was crushed to 40 mesh (0.6 mm.) and roasted. The calcines were leached with 8 to 10 percent sulphuric acid solution. There was thus extracted 50 percent of the copper and a small amount of nickel. The dried residues were mixed with charcoal, fused, and cast into anodes. These anodes contained Ni, 79.8 percent; Cu, 18.0 percent; Fe, 2.0 percent. The anodes were 5½ x 5½ inches (14 x 14 cm.), and had two lugs which were drilled and tapped for copper rods. Each anode weighed approximately 1900 grammes, and two anodes were used in each tank.

The cathodes were sheet lead or sheet aluminum 6 x 6 inches (15 x 15 cm.) There were three in each tank, suspended in a bag of light canvas which was pulled over a wooden frame in order to keep it distended.

The electrolyte was a solution of nickel sulphate usually kept at from 5.5 to 6.5 percent nickel (110 to 130 lb. (50 to 60 kg.) nickel per fluid ton). The temperature of the electrolyte was maintained between 40° and 50° C. In starting a new tank 3.5 percent (70 lb. per fluid ton) of ordinary calcite crushed to 200 mesh (0.125 mm.) was added to the electrolyte. The solids were kept in suspension by circulation by a pump which discharged into a small lead tank resting on a hot plate, and from

which the electrolyte flowed by gravity back to the stoneware tank. Limestone at 200 mesh (0.125 mm.) was added hourly to the electrolyte. The amount added was based on the rapidity of solution of the anode. Calculated back to the original matte the quantity of lime-rock used ranged between one-half and three-quarters of a ton of limestone per ton of matte. Necessarily, more limestone was needed than the equation called for. A small amount of glue was used in the electrolyte.

A current density of 13 amperes per sq. ft. (143 A. per sq. m.) of cathode gave excellent nickel. The potential required was 2 volts. The ampere efficiency at the cathode was 98 percent, at the anode sometimes slightly over 100 percent.

In practice there would be removed regularly from circulation a portion of the accumulated mud, in order to keep the percentage of solids constant. This mud, after washing, showed on drying Ni, 8.28 percent; Cu, 9.10 percent; S, 8.88 percent; CaO, 30.1 percent; Fe, 1.4 percent. Fusions were made of this mud with silica as flux, in graphite crucibles. The product was a slag: SiO_2, 48 percent; CaO, 38 percent; and a matte containing Ni, 36.1 percent; Cu, 37.8 percent; S, 19.5 percent.

The procedure suggested is to roast this matte, then reduce and cast it into anodes. These anodes will be used in copper refining tanks using the copper-rich acid liquor from the leaching operation. Good cathode copper can be produced until the copper content of the electrolyte is reduced to 1.5 to 1.7 percent copper. Not enough of this mud was obtained to furnish material for such anodes in a copper tank. Anodes made from other sources were used. A current density of 12 amperes, at 0.35 volt, was used. Insoluble lead anodes were used after the copper was reduced to 1.47 percent. The potential for the same current was now 2.2 volts. From 1.47 to 0.054 percent Cu, the ampere efficiency was 66.2 percent, during the final reduction to 0.0115 percent Cu the ampere efficiency was 46.7 percent.

The copper department in such a refinery would receive the leach liquor from the roasted matte; would electrolyze this with anodes made from the mud in the nickel department. When the copper content of the electrolyte reached 1.5 percent, the solution would be electrolyzed with lead anodes to remove most of the

copper. It would then go to evaporating pans and crystallizing tanks. The mother liquor containing the acid and the rest of the copper would be used for further leaching. The crystals would go to the nickel department.

The crude nickel anodes would be received in the nickel house. Electrolytic nickel would be produced, and a lime mud which would carry away a chemical equivalent of nickel. This nickel would be returned from the copper house as nickel sulphate. In addition to this there is required an amount of nickel as nickel sulphate, the chemical equivalent of the copper in the anodes, since this amount of nickel is plated at the cathode while the copper in the anode is being dissolved. The copper house is required to supply then 117.4 lb. (62.5 kg.) of nickel as nickel sulphate for every 63.5 lb. (28.8 kg.) of copper in the anodes. To obtain this some of the nickel house anodes may be electrolyzed in the copper house, to supply any nickel sulphate not made up by the amount of nickel leached out from the roasted matte. The amount of acid required is similarly twice the chemical equivalent of the copper in the nickel-house anodes.

The rare metals platinum and palladium, which occur, do at present prices form an important constituent[3] of these mattes. These metals are retained in the slime which forms an adherent crust on the anodes, and does not tend to fall off. The slime, if desired, may be obtained for the most part free from the lime mud.

No laboratory work was done using a salt-cake fusion. Anodes made after such a fusion might be taken as having from 2 to 5 percent copper. The amount of mud made would be correspondingly small, and this might be returned for salt-cake fusion with the original matte. The nickel sulphate required would come from the electrolytic refining of the copper made from the "tops."

Nickel cathodes free from iron and containing as little as 0.01355 percent copper were made. A small amount of unprecipitated copper was always found in the electrolyte. It was always very small, often only 0.001 percent Cu. The double basic sulphate is easily filtered, *i. e.*, it is easily retained on any filter paper. Bags of light canvas were found more satisfactory than heavy material.

[3] Ontario Nickel Commission report, p. 486.

DISCUSSION.

CARL HERING[1]: In this process a powdered material is suspended in the electrolyte. I made some experiments quite a number of years ago[2] under the microscope to study the migration of solid particles in electrolytes and found that they traveled very much like the atoms themselves. They always had a definite direction; they would either always go to the anode or always to the cathode; most of them, however, traveled with the current, that is, to the cathode, and they seemed to travel at a speed proportional to the electromotive force. Sometimes they traveled so fast that they would pass over the field of the microscope so quickly that they could not be seen; it required an extremely low electromotive force to be able to see them travel.

This seems to me to be important in a case of this sort in which a foreign material is suspended in an electrolyte. It should be remembered that such a material is going to travel with (or against) the current too, and it is therefore likely to be deposited on the cathode with the purified material; therefore in all such cases it seems to me to be necessary to put a porous cell around the cathode; I notice that in the present case the author puts a canvas bag around it, but the meshes of this I should imagine would be too large to prevent those little particles from going through. Another illustration of the deposit of such foreign materials is in case of a silver voltmeter, with which very accurate measurements were made; it was sometimes found that the deposit was greater than 100 percent. That was probably due to the fact that solid particles traveled with the current and deposited on the cathode.

C. P. MADSEN[3]: In absence of having seen tests on the process I cannot make any comment, but would like to ask the author why glue was added to the solution.

CARL HERING: I do not know, I am not the author, I can not answer. I do not know whether glue would travel like a solid particle or not. Those studies I made were with solid particles

[1] Consulting Elec. Eng., Philadelphia.
[2] Trans. Am. Electrochem. Soc. (1906), **10**, 35.
[3] Consulting Eng., New York City.

which could be seen in the microscope; I do not know how glue would act.

J. W. RICHARDS[4]: Evidently the glue was put in for the purpose of solidifying the nickel deposit, and I think Dr. Hering's criticism that the canvas would not be sufficiently fine to keep these particles from traveling with the current to the cathode is answered by the fact that there was not over a thousandth of one percent of impurity in the cathode metal, which shows that the canvas bag did entirely prevent any mechanical contamination of the cathode by suspended particles.

CARL HERING: That does not mean that the solid material, the calcium carbonate, did not travel. It would appear on the cathode, not as a metal, but as a compound.

GEO. B. HOGABOOM[5]: The presence of glue in nickel materially changes the character of the deposits, making it very dark and brittle.

G. A. GUESS (*Communicated*): In reply to Dr. Carl Hering, I wish to say that lime particles did not penetrate the light canvas bag, much less adhere to the cathode. These experiments were carried on for several months and many hundreds of analyses were made. As stated in the paper, bags of heavy material were not satisfactory. Glue was added in small amount as it appeared to produce a better looking cathode. If too much glue were used the nickel would be dark colored.

[4] Prof. of Metallurgy, Lehigh University.
[5] Electroplating Advisor, U. S. Bureau of Standards.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

ELECTRO-PLATING ON IRON FROM COPPER SULPHATE SOLUTION.[1]

By Oliver P. Watts.[2]

Abstract.

A discussion of getting good electro-platings of copper upon iron by preliminary dipping of the iron article in solutions of various metals which lie, electrochemically, between iron and copper. Arsenic, antimony, bismuth, lead and tin dipping solutions were tried, and arsenic, lead and antimony solutions found effective in securing a good subsequent electro-plating of copper. Even bismuth can be electro-deposited on iron by using a preliminary arsenic or antimony dip. Nickel can be deposited on aluminum by using a ferric chloride dipping solution; the author was not able to improve on this. [J. W. R.]

For electro-plating on iron or steel from a copper sulphate solution, platers have in the past found it necessary either to give the metal a preliminary coat of nickel or to plate it with copper from a cyanide solution before transferring the object to the copper sulphate plating bath. Since the sulphate solution is much more satisfactory than the cyanide for the production of heavy deposits of copper, where these are required the plater has been compelled to maintain two different plating solutions, and to perform two distinct plating operations. The desirability of being able to plate directly on steel in the copper sulphate solution is apparent, and many persons have tried to discover a method of doing so, but until recently in vain.

The difficulty which proved insuperable in the many years that copper plating on iron has been practiced is the attacking of the

[1] Manuscript received February 3, 1919.

[2] Associate Professor of Chemical Engineering, University of Wisconsin.

steel by the electrolyte, and the deposition of copper "by immersion." When a strip of zinc or iron is dipped into a solution of copper sulphate, metal dissolves and a chemically equivalent amount of copper is deposited from the solution. This action is in its nature the same as the dissolving of a metal by sulphuric acid, differing only in that copper is plated out of solution instead of hydrogen. The trouble in plating upon a metal which receives such a deposit by immersion is two-fold: when reaction between the metal and the solution is rapid the character of the deposit is the same as that obtained in extremely rapid deposition by the current, it is rough, powdery, and non-adherent; and since the driving force which causes the metal to be deposited is the going into solution of an equivalent amount of metal from the object which is being plated, there is a continual undermining of the foundation upon which the deposited metal rests, so that the plating blisters or becomes entirely detached after a short period of service. The remedy which naturally suggests itself is to use a large current at the outset, and cover the object with an electrolytic deposit so quickly that there will not be time for the corrosive action of the electrolyte to do serious damage. This method is successful when the rate of attack by the electrolyte is slow, *i. e.,* when the E.M.F. between the object and the metal constituting the deposit is small, but when this E.M.F. is great, even starting in, or "striking," at a high current density fails to overcome the trouble. It is for these reasons that platers have found it impossible to secure from neutral or acid solutions of their common salts an adherent electro-plate of copper, silver, platinum, or gold on such electro-positive metals as magnesium, zinc, or iron.

At the annual convention of the American Electro-platers' Society held at Detroit in July of last year, a paper was presented by John Satka, of the Chicago Branch,[3] which gave a method of copper-plating on iron or steel from a copper sulphate solution without the usual preliminary plating in the copper cyanide or the nickel solution. The method consisted in immersing the cleaned steel in a solution of 8 ounces of arsenious oxide to a gallon of hydrochloric acid (60 g. per liter), rinsing, and plating in the copper sulphate bath at a current density of 10 to 15 am-

[3] August issue of Monthly Review of American Electroplaters' Society.

peres per square foot (1.10-1.65 A. per sq. dm.). Care should be taken that contact with the cathode rod is made as soon as the steel touches the solution. Since it is necessary that the arsenic dip shall reach every portion of the surface of the steel, the articles should be dipped in the solution several times to insure complete substitution of the rinse water by the arsenic dip. Under these conditions a bright copper coating is deposited, which will withstand polishing, scratch-brushing, and ball-burnishing. This method is already being used by a number of platers, and seems destined to wide usefulness.

In seeking a reason for the remarkable effect produced by immersion of iron in the arsenic solution, one naturally thinks of the prevention of the corrosion of iron by hydrochloric or sulphuric acid that is brought about by dissolving a little arsenic in the acid. Although a striking coincidence, this other effect of arsenic furnishes no key to the mystery under consideration. Arsenic prevents corrosion of iron by providing a cathode material on which the potential required for the evolution of hydrogen exceeds the potential of iron, so that iron is unable to displace hydrogen from the solution, and corrosion of the iron ceases;[4] but when plating on iron from copper sulphate solution the trouble heretofore experienced has been due to the displacement of copper, not hydrogen, and the ability of arsenic to prevent the displacement of hydrogen by iron seems to offer no explanation of the beneficial effect of its presence in this case.

During immersion in the arsenic dip used by Mr. Satka, a coating of arsenic, more or less perfect, must be deposited on the iron, and it is the presence of this coating of arsenic that makes it possible to secure an adherent deposit of copper. But how? Heretofore deposits produced by simple immersion have been regarded with suspicion by the plater; if thin, they do not protect the iron, but rather accelerate rusting, and if an attempt be made to secure a thick deposit by prolonged immersion of the iron, the entire deposit is liable to come off. Does this film of arsenic differ in some fundamental way from all other deposits produced by immersion, or can its beneficial effect in plating on iron with copper be brought about by other metals? Experiment alone can answer this question.

[4] Trans. Am. Electrochemical Society (1912), **21**, 337; (1917), **32**, 257.

The only metals which give any promise of furnishing a substitute for arsenic are those which are below iron, and are above or but a trifle below copper in the electrochemical series. The other metal must be below iron in order to deposit on iron by immersion, and should be above copper or but little below it, else the remedy will prove worse than the disease which it is hoped to cure. According to Neumann, the common metals which lie within these limits are the following:

	Sulphate	Chloride
Iron	+0.093	+0.087
Nickel	—0.022	—0.020
Tin	....	—0.085
Lead	....	—0.095
Bismuth	....	—0.315
Antimony	....	—0.376
Copper	—0.515	....
Arsenic	....	—0.550

Measurements, by the writer, of the E.M.F. between copper and copper plated with arsenic, antimony, and bismuth, gave values of 0.034, 0.068, and 0.042 volts, respectively. All three metals were positive to copper, but only bismuth precipitated copper from the acid sulphate plating bath used in the experiments which follow. The failure of antimony to do so is probably due to the formation of an insoluble basic sulphate.

Of the metals in the list above, nickel lies too close to iron to yield a deposit by immersion, hence none of its salts were tried as dips for iron, but solutions of each of the others were used as preliminary dips before copper-plating.

It is of course essential that the coat of metal obtained by immersion shall be smooth and fairly adherent; much trouble was experienced in obtaining such a deposit with some of the above metals.

THE ARSENIC DIP.

In using the arsenic dip the concentration of arsenious oxide was varied from 10 to 60 grams per liter of hydrochloric acid, and the acid was also diluted with two volumes of water without detriment to the effectiveness of the dip. Good results in copper-plating were obtained with a variation of the time of immersion in the arsenic dip from 10 seconds to 6 minutes. Several specimens failed to take a perfect coat of arsenic, as was shown by

removal of the copper plate from portions of the sheet by the scratch brush. The reason for these failures was not apparent, as the sheet iron had been put through the usual cleaning process, *viz.,* removal of mill scale by hydrochloric acid, rinsing, brushing with a brass scratch brush to brighten the surface, dipping for an instant in the pickling acid, rinsing, immersion in the arsenic dip, rinsing, and plating. Sulphuric acid was tried as a solvent for the arsenic instead of muriatic, with only slightly less satisfactory results.

THE LEAD DIP.

A lead dip was prepared by diluting 40 c.c. of the lead fluosilicate plating bath with 400 c.c. of water, adding 100 c.c. of fluosilicic acid, and a solution of 4 grams of gelatine. This acted slower than the arsenic dip, and required about 30 seconds' immersion to insure a good copper plate on iron. There were less failures from bad spots in the copper plating than with the arsenic dip. Solutions of lead nitrate acidified by nitric acid and of lead acetate made acid by acetic acid were also tried, but the copper plate could be rubbed off in every case in which these dips were used. Their failure as dips is probably due to the crystalline nature of the lead deposited from them.

THE ANTIMONY DIP.

In experimenting with antimony chloride a very concentrated solution of magnesium chloride, strongly acidified with hydrochloric acid, was used as a solvent, in order to lessen the amount of hydrochloric acid which would otherwise have been necessary to keep the antimony in solution and to prevent a black, spongy deposit of antimony on the iron. About 1 c.c. of formalin was added per 100 c.c. of solution, to diminish attack of the iron by the hydrochloric acid. In order to obtain uniformly good deposits of copper by the use of this dip it was found necessary to acidify the copper plating bath more strongly than when using the arsenic and lead dips.

THE BISMUTH DIP.

From solutions of bismuth chloride it proved very difficult to get a smooth, bright deposit on iron by immersion. In spite of the use of magnesium chloride and hydrochloric acid in the dip,

a trace of water on the iron when immersed would cause a black smut to be deposited. By dipping the iron in concentrated hydrochloric acid just before immersing in the bismuth dip some fair-looking coats of bismuth were obtained, but except in the case of two narrow strips of iron, a good copper plating could not be obtained over the bismuth; the copper solution seemed to get through the bismuth coating and attack the iron, for the copper plate was easily rubbed off. Deposition of copper on bismuth by immersion was found to take place slowly, and to give a coating of copper which stood severe scratch-brushing.

THE TIN DIP.

An acidified solution of tin chloride seemed to give a good deposit of tin on iron by immersion, but it was impossible to obtain a good plate of copper after using the tin dip, although copper deposits on tin rather quickly by immersion. A good copper plate was obtained on a bar of tin. The reason for the failure of the tin dip is not clear.

THE COPPER DIP.

Success with the antimony, arsenic, and lead dips suggested an attempt to copper-plate iron by the use of a copper dip made up on similar lines, *i. e.,* a solution weak in copper and highly acidified. It was found that increase of acid in a dilute solution of copper sulphate slows down the rate of deposition by immersion on iron, and with a solution of copper sulphate in concentrated sulphuric acid the deposition of copper becomes very slow. All attempts to plate upon such deposits failed.

The finding of several other metals which may be substituted for arsenic as the basis of a dip for the plating of iron from a solution of copper sulphate, seems to indicate that the beneficial action of arsenic consists merely in its forming a coating over the iron which adheres fairly well, and protects the iron beneath for the brief time, 4 or 5 minutes, needed to give it a coat of copper by the current. To confirm this view it seemed desirable to try the arsenic dip as an aid to plating with some other metal than copper from a solution that does not give an adherent deposit on iron. For this purpose bismuth was selected as the substitute for copper.

PLATING ON IRON WITH BISMUTH BY AID OF THE ARSENIC DIP.

Fifty grams of bismuth oxide was dissolved in 100 c.c. of concentrated hydrochloric acid, to which 10 c.c. of formalin was added to minimize attack of the iron by the acid. When tested by plating on copper, a gray, adherent deposit was obtained. On iron the deposit by the current peeled in spots at the end of five minutes, and a scratch brush removed most of what remained. However, both the arsenic and antimony dips made it possible to get a good, adherent plate of bismuth on iron. Use of the lead dip resulted in the bismuth plate taking the form of a black smut. An attempt to plate on sheet lead with bismuth showed the failure of the lead dip to be due to attack of the lead by the plating bath, with deposition of the smut. Owing to the brittleness of bismuth, it cannot be recommended as suitable for general use in plating.

SILVER-PLATING BY AID OF THE DIPS.

Acid solutions of silver salts give a spongy deposit on iron by immersion, and plating from them is therefore a failure on iron. Tests of the above dips as aids to plating on iron from an acid solution of silver fluoborate resulted in non-adherent deposits in every case. This is not surprising in view of the distance that silver is below arsenic, antimony, and lead in the electrochemical series. On dipping these three metals in the silver-plating solution a black, spongy deposit instantly formed on each metal. This failure of the dips for silver-plating made their testing for plating with gold and platinum, metals of still lower potential, unnecessary.

THE ARSENIC DIP FOR COPPER-PLATING ON ALUMINUM.

One of the most successful methods of plating on aluminum, a problem that long baffled the plater, consists in dipping the cleaned aluminum for a half minute into dilute hydrochloric acid which contains 1 percent of ferric chloride, and then plating with nickel. The beneficial effect of this treatment is believed to lie in the deposition of iron by immersion, to which metal the nickel plate adheres well. If a perfect coating of arsenic, antimony, or lead could be given to aluminum by the dips previously employed, it should be possible to plate aluminum with copper instead of nickel, which, for obvious reasons, it is now necessary to use.

On immersing aluminum in the bismuth and tin dips, a black smut at once formed, so these were abandoned. Several samples of copper plate on aluminum were prepared by aid of the arsenic and lead dips; they stood polishing, but on cutting away the edges of the sheet and stripping off the copper plate, adhesion seemed slightly poorer where the sheet had been immersed in the dip than it was above this point. No beneficial effect was observed for any of the dips used.

In the experiments in copper-plating on iron, flat sheets, 2 by 6 inches (5 by 15 cm.) were employed; it is therefore impossible to draw conclusions regarding the effectiveness of the dips on the large and irregular material that frequently must be plated in commercial work. Objects with deep recesses, into which the copper sulphate solution will not "throw" well, may be expected to cause trouble in using the arsenic dip, and may prove impossible to plate by this means. Although adhesion was good enough to stand polishing, it is inferior to that of the ordinary electroplate, in the production of which deposition by immersion has played no part.

Some misanthrope, who would discourage human endeavor, once said "There is nothing new under the sun," and a study of the history of the most revolutionary discoveries in science seems to bear out this statement. In spite of the novelty of the arsenic dip, and the accomplishment by its use of that which had been regarded as impossible, the principle underlying its action has long been employed by platers in the "blue" or "quick" dip. This consists of a salt of mercury dissolved in potassium cyanide, into which articles of copper or brass are dipped before silver-plating. Its purpose and method of operation are exactly those already set forth in explanation of the arsenic dip; mercury is deposited on the other metal by immersion, and lowers the potential of the object to be plated so that corrosion by the electrolyte is diminished, and in consequence there is a slightly better adhesion of silver plate when the "quick" dip is used. Since the potentials of copper and brass in dilute cyanide solutions are not much greater than that of silver, the attack of these metals by the silver bath is not severe, and a very satisfactory silver plate can be secured on copper or brass without employing the "quick" dip.

The arsenic dip is merely the latest case in which the plater's

old enemy, deposition by immersion, has been made an ally by the simple expedient of introducing an intermediate stage—the deposition by immersion of a metal that is only part way down to the potential of the metal with which it is desired to plate the electro-positive metal.

SUMMARY.

It has been discovered recently that by immersing iron for a few moments in an acidified solution of arsenious oxide, an adherent copper-plate may be deposited from an acid electrolyte, but no adequate explanation had been offered for this unique effect of arsenic. In this paper it has been shown that certain solutions of lead and antimony may be substituted for the arsenic dip, previous to direct-current plating of copper on iron from copper sulphate.

It appears to be impossible to obtain a perfect plate on iron from solutions of bismuth chloride by the usual methods of electro-plating; but use of the arsenic or antimony dip is attended with the same successes as in copper plating.

The successful substitution of solutions of antimony and lead for arsenic, and the application of these dips to plating on iron with bismuth, show that the beneficial effect of the arsenic dip is not due to a property peculiar to arsenic alone, but is the result of coating the iron with a metal whose potential in acid solutions is so near to that of copper that it is possible to deposit a good copper plate upon it, yet whose potential is not so far below iron that it will deposit on iron in a powdery, non-adherent form.

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DISCUSSION.

W. D. MAINWARING[1] (*Communicated*): One of the problems upon which I was consulted recently was the electrodeposition of nickel upon steel tools. The particular articles were surgical chisels for government uses.

Upon first being consulted, the question was asked, are you

[1] Production Eng., Cleveland, Ohio.

giving the tools a preliminary copper plate, the answer returned was no. An investigation was next conducted to find out if possible what kind of surgical tools were double plated. I was greatly surprised to find that practically all such articles were nickel plated directly upon the instruments.

The tools themselves were next examined and it was found that the nickel plate was peeling from them in various manners, at the point, along the edges and down the sides. The next step considered was the reasons for the peeling of the nickel. Every factor was considered in the problem, the thickness of coating, on point and sides, the effect of honing the tool after plating was completed. This operation had to be done at this stage in order to have cutting edges in excellent shape. It was contended that no matter when the honing was done the nickel-plating should stick. During the examination of the tools, it was found that every tool on which the nickel plate was stripping had a film of dirt and oxide between the plate and surface of tool.

This report relating to the above caused more care to be taken in cleansing the tools, but still the thing kept occurring. It had been observed that the tanks were uncovered and anodes and surface of liquid were very dirty, this was also pointed out. Better results were finally obtained, but the percentage of waste remained extremely high.

Since working upon this problem, an effort has been made to arrive at an answer to the question, how did the dirt get between the nickel-plating and surface of the tool, the tools being perfectly clean when placed in the tank. It has been stated that the surface of tanks and anodes were extremely dirty. When the articles to be plated are placed in a tank and the current is turned on, an active decomposition of the water at once takes place, a flow of water to the surface of the tool immediately begins to make up for that decomposed, any dirt that would be on the surface of the tank would be drawn down and deposited with the nickel, just as long as the tools remained in the tank.

I might say that the particles that covered the surface of the tank were not all dirt; some of the particles were salts, nickel salts that had crystallized out; they were floating on the surface of the tank, and also the anodes. This was pointed out. In these tools after the tool was honed, just breaking the nickel, the entire

nickel plate would strip off and you could see the flow of dirt from here down the tool, and it all settled in the point.

This discussion is written to emphasize the point that in any method of plating used, the fundamental principle of success is cleanliness, not only of tools but tanks; they should be thoroughly cleaned out at times and not allowed to become covered with dirt and dust.

It seems to me that all kinds of steel articles should receive a preliminary coating of copper before the nickel-plating is done. The process described by Mr. Watts seems to give an efficient method for doing this.

GEO. B. HOGABOOM[2]: In the gentleman's discussion, I think his argument in regard to the dirt on top of the solution going down into the solution and depositing on the work is equally applicable, whether the article was copper plated or not copper plated. And copper plating would not prevent the dirt from appearing.

The question of depositing nickel upon steel directly is influenced more by the amount of current that is used. You can get a current that would deposit hydrogen primarily, or you can get a current that would deposit hydrogen with the nickel. In the first place, you would have a distinct peeling; in the second place, you would have a peeling at the edges and pitting. It is not a question of cleanliness. If the solution had some crystals of nickel sulphate on the anodes it would show that there was something radically wrong with his nickel solution. If he was using a double sulphate solution he would not be able to use over 12 oz. per gallon (90 gm. per liter), and if the temperature of the room should decrease to 50° F. (10° C.), there would be crystallization of the salts, but, if a single sulphate solution was used, that would eliminate crystallization.

There should not be, nor is, any difficulty in depositing nickel directly upon steel. Shears are plated with nickel without previous copper plating. After they are plated, they are ground, that is, the nickel on the inside of the shear is ground off, and the edges are sharpened. The adherence of the nickel depends entirely upon the condition of the solution and the amount of

[2] Electroplating Advisor, U. S. Bureau of Standards, Washington, D. C.

current used, and the same thing should be true for surgical instruments.

W. D. MAINWARING: During the working out of that problem we tried all the current densities, and also the question was taken up with certain physicians in Philadelphia as to the advisability of first plating the article with copper. We did not plate the tools with copper, but we finally succeeded in getting a coating of nickel to stick.

A. G. REEVE[3]: We plated our surgical instruments with nickel. The plating may perhaps have peeled from a few of them, but I think the plating held pretty well on nearly all, and we did not plate any of them with copper. We plate a great deal of our cutlery and deposit nickel on much of it, not on all; on some we deposit silver directly on the steel, and I heartily agree with what Mr. Hogaboom has said about causing the nickel to adhere directly to the steel. The idea of the dirt being drawn from the surface of the tank is a new one to me. In all my experience I never saw anything that gave me the idea that that could take place. Perhaps it does; but I have noticed this, in passing a well-cleaned steel article from a rinse-tank into a nickel-plating bath, if there is dirt or anything on the top of the bath, the film of liquid covering the articles to be plated will spread immediately when it strikes the surface of the bath, as the work enters, so that the dirt that was there in minute floating particles immediately spreads out, making a clean place through which the work passes and is clean after it gets into the bath. Of course, if the dirt, by some means or other, is drawn down on to the work after that, harm will be done, but, for myself, I never saw any work which I thought was caused to be defectively plated in that manner. We have found, however, that holding the work up and looking at it as it is coated with water and observing the water-break, as we call it, to see if the water will run off or stand on the article, in drops, is quite an inadequate test for determining the cleanliness of steel. Steel used in cutlery frequently has more or less numerous very minute pinholes which can only be observed with the aid of the microscope and are absolutely invisible to the naked eye, and these reach in to a sufficient depth

[3] Res. Dept., Oneida Community Ltd., Kenwood, N. Y.

and seem to be capable of holding enough foreign materials, grease, buff dirt, pickling acid, or most anything of that kind, so that when you hold the article up to observe as to its cleanliness, you possibly do not get a water-break because the main surface is absolutely clean, and the water cannot break away from it, but sufficient foreign matter exudes from the pinholes to prevent deposition immediately adjacent thereto, and the imperfection is not bridged over, so that the water-break is very inadequate as a test for the degree of cleanliness required by our line of plating. Our experience has been that if we get the work properly and thoroughly cleaned, and, as we say, cleaned to a sufficient depth, nickel plating, if put on as Mr. Hogaboom says, with sufficiently low current density, will adhere with great tenacity and can not be pried off by mechanical means.

C. P. MADSEN[4]: I regard this paper as a very important contribution to the art of copper deposition on ferrous metals, I, however, question the suggested value of the process outlined in this paper as a means of copper plating iron before nickel plating. So long as metals are deposited in the porous condition shown by Dr. Watts' former paper entitled "The Protection of Iron by Electroplating," the advantage of first plating the iron with copper is questionable, especially for such articles as cooking vessels and surgical instruments, whether the copper deposit made by these new processes of treating the iron surface is more impervious than that made from a cyanide bath or not, because verdigris can still form through the porous nickel deposit.

I agree with Mr. Hogaboom that nickel can be deposited directly upon iron at least as adherently as copper, and that, therefore, an intermediary coating of copper is not necessary for this purpose.

The origin of the dirt which Mr. Reeves refers to may not be so mysterious when the character of commercial nickel anodes is understood. These anodes not only contain considerable iron but also many insoluble impurities. Under certain conditions a considerable quantity of nickel oxide in the form of a black insoluble powder is also formed. While normally this oxide and insoluble impurities should settle in the bottom of the bath it can under certain conditions be carried across the cathode, which may account for the dirty surface mentioned.

[4] Consulting Engineer, New York City.

I may also add that I do not quite agree with Mr. Reeves that dirt in the bath is the only cause of pinholes. It is true, of course, that any kind of impurity in a deposition bath causes imperfections in the deposits, but I have also encountered pinholes in deposits made from chemically pure and clean baths and have, therefore, concluded that they may be an inherent result of certain factors of deposition.

WM. BLUM[5]: It is interesting to note that Dr. Watts (as well as Mr. Satka, the author of this method), was able to obtain good deposits of copper upon steel covered with a preliminary coating of arsenic, a metal usually placed below copper in the electrochemical series; but was unable to substitute for the arsenic certain of the metals which are intermediate between iron and copper. This behavior illustrates the fact, emphasized a few years ago in a paper by Dr. Frary, that the question as to whether and in what form a metal will deposit upon another metal by immersion, will depend upon the relative potentials between the two metals and the particular solution employed. This relation in turn will depend not so much upon the normal position of the elements in the electrochemical series, as upon the character, and especially the metal ion concentration of the solution.

J. W. RICHARDS[6]: In regard to the use of nickel plating on surgical instruments, it is interesting to note that one of our members, Mr. Haines, manufactures an alloy called stellite, which is a substitute for tool steel and makes an excellent material for surgical instruments. I understand that a large amount of his activities during the war has been in the manufacture of surgical instruments out of stellite, that being one metal that is not attacked by solutions.

GEO. B. HOGABOOM: Why would not chromium steel act as well? I have some knives on my table made of chromium steel which have been used five years, and you can cut any kind of fruit with them and leave the juice standing on them over night or any length of time, and they are absolutely free from stains.

J. W. RICHARDS: I think one question is that of resisting the antiseptic solutions which are used and may attack the chrome steel but are said not to attack the stellite.

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[6] Prof. of Metallurgy, Lehigh University.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 3, 1919, President Tone in the Chair.

REMARKABLE PITTING OF ELECTROPLATING[1]

By Oliver P. Watts.[2]

Abstract.

When making some lead platings, heavy pitting suddenly and irregularly appeared. After considerable study of the cause, it was finally found to be due to air dissolving in the electrolyte while it was resting over night and cooling, which was then expelled as minute air bubbles on the work when the bath was heated up by the passage of the current. The microscopic bubbles clinging to the work, which had been left immersed in the bath, started the pitting. [J. W. R.]

Although the conditions which led to the unusual pitting which is the subject of this brief paper are very seldom met with in plating, the possibility that others may encounter the same difficulty makes it seem desirable to describe the phenomenon, and to tell its cause.

The trouble occurred in giving steel articles a plating of lead 0.010 inch (0.25 mm.) thick from a solution of lead fluosilicate, and consisted in the lead coating being more or less completely perforated by hundreds of small holes, which, from their size, may best be called pin-holes, and which of course rendered the lead plating useless.

A peculiarity not shown in the accompanying picture is the tendency of the pits to appear first along a sharp edge, *e. g.,* where a line of slag had been removed by the pickling solution. The most surprising thing about this attack of pin-holes was its suddenness and severity. Plating had been going on for six weeks

[1] Manuscript received February 3, 1919.

[2] Assistant Professor of Chemical Engineering, University of Wisconsin.

before the trouble developed, but at its first appearance a half day's output of the plating tanks was spoiled. The trouble then grew less, and soon ceased. Several days later it recurred, all plating from the three tanks being badly pitted. Plating was then stopped until the cause of trouble could be determined and a remedy found.

There was much mineral oil on the articles as received at the plating room, and some of this was found to be carried over from the electric cleaner employed to remove it into the acid pickle used to remove rust just previous to plating. The hydrogen set free from the acid broke this up into microscopic drops which were found floating on the surface of the pickle, through which the work must be withdrawn. Complete elimination of the oil, however, did not stop the pin-holes.

The observation that many of the holes extended only part way through the lead coating furnished the clue to the cause of the trouble, *viz.,* that the holes were due to the collection of bubbles of air which was being slowly expelled from the plating bath.

During the hot weather of July and August, when the temperature of the solution was 115 to 120° F. (44 to 47° C.) and the tanks were in use 24 hours a day, this trouble did not occur; but the coincidence of the cool nights of September with an insufficient supply of material to be plated, so that the tanks occasionally stood idle from 8 P. M. to 9 or 10 A. M., permitted the solution to cool from its working temperature to about 78° F. (25° C.); this cool solution then dissolved a considerable quantity of air. When the current of 900 amperes was turned on, four or five hours were required for the solution to regain its normal working temperature; during this time the dissolved air was slowly expelled and collected in minute bubbles, which clung to the articles that were being plated and so caused the pin-holes. A similar expulsion of dissolved air is seen when a tumbler of cold, aerated water is brought into a warm room; bubbles of air slowly gather on the sides of the tumbler. The large current used, the high resistivity of the solution in comparison with the copper sulphate bath, for example, the small surface exposed to the air (the tanks were 28 inches (70 cm.) deep and slightly less in width), and the heat insulation furnished by the 3-inch (7.5 cm.) plank of

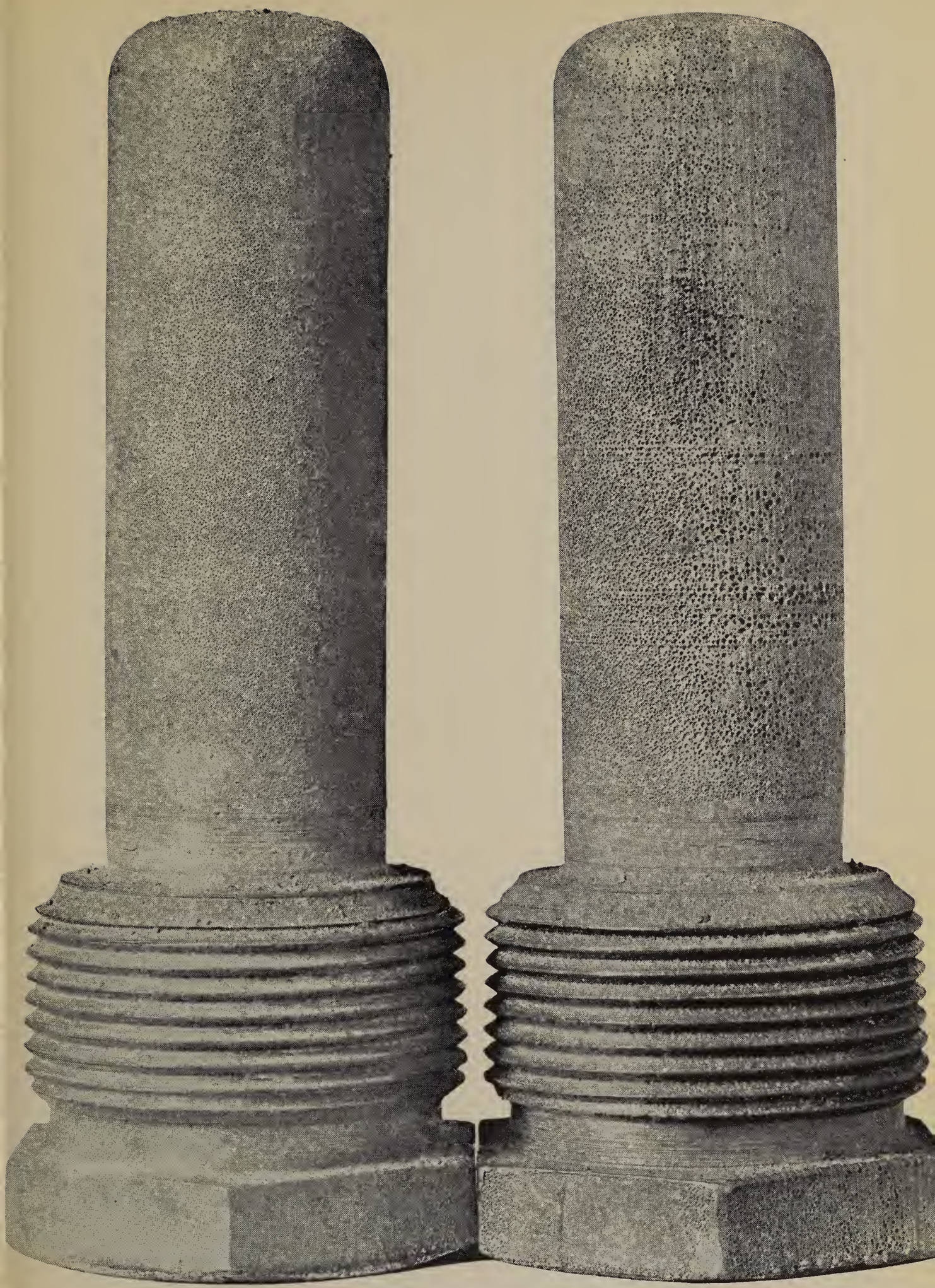

Fig. 1.

which the tanks were constructed, all contributed to the trouble since their combined effect was to make the operating temperature of the solution over 20° F. (11° C.) above that of the room. The three hours required for deposition gave ample time for bubbles of air to collect.

The difficulty was overcome by heating the solution to a little above its working temperature by means of a lead steam coil before resuming plating, whenever the tanks had been out of use for any considerable time.

Similar trouble by gas-pitting may be expected whenever a plating solution that is used for heavy deposits, and operates at an elevated temperature, stands idle and cold for many hours; but if there is a steady evolution of hydrogen from the cathode during normal operation of the bath, as with the cyanide copper or the brass solution, no trouble need be anticipated, for the stream of gas, ascending along the surface of the metal, will prevent bubbles from clinging long enough to form pits in the plating. Pitting may be expected only when the rate of evolution of gas from the work is extremely slow.

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University of Wisconsin.

DISCUSSION.

GEO. B. HOGABOOM[1]: Dr. Watts' paper opens up a discussion upon electroplating that I think is more broad than his paper would indicate. If you read any text book upon electroplating, one of the first postulates is to boil a new solution. In looking up some of the literature I find that Roseleur in 1852, was one of the first to recommend the boiling of solutions, as it would improve the deposit. It is quite evident that the boiling of the solution is to drive out whatever air there may have been in the water in mixing the solution.

Several years ago I was working on heavy silver deposits and after crystallizing out the carbonates by lowering the temperature of the solution they were filtered off. To each 100 gallons of

[1] Electroplating Advisor, U. S. Bureau of Standards.

solution 20 gallons of very cold water directly from the faucet were added. I was required by the President of the Company to immediately place some work in that solution and run it. The deposits were run for twenty hours. Every piece that was put into the bath came out in exactly the same pitted condition as Dr. Watts' booster shell. That solution was then run with the porous cell for five hours and work put in it and the deposit became normal. The following day I tried it on another bath with some work and duplicated the condition and results, in fact, it was duplicated the day after. A year later it was tried again with the same pitting. It was evident that the pitting was due to the addition of cold water which had sufficient air in it to cause the pits.

A lecture presented at the Thirty-fifth General Meeting of the American Electrochemical Society at New York City, April 3, 1919.

THE OXIDATION OF AMMONIA.

By W. S. Landis.[1]

Just when ammonia was first intentionally converted into nitric acid or nitrate products, I have been unable to learn. Soil bacteria have been doing it for ages. Since the invention of gunpowder, with its increased demand for nitrates, man has endeavored in places to assist such bacterial action and thereby lighten his task of recovery. The direct oxidation of ammonia by air through chemical or electrical action seems to have been a scientific curiosity more than a hundred years ago. Eighty years ago Kuhlman, a French technical chemist, after describing at length some very interesting experiments of his own, concluded with the significant remark, "While under present conditions the transformation of ammonia into nitric acid with the assistance of platinum and air has no great value, yet the time will come when it may be of commercial importance."

This commercial prophecy, presented so many years ago, has been a long time reaching fulfillment. Ten years after its utterance, when Chilean nitrate was reaching Hamburg in quantity and prices of nitrate began to decline, its fulfillment must have looked hopeless, as well as during the long years when ammonia prices and nitrate prices were strictly competitive and there was no margin of difference to pay for the transformation. Nevertheless, in the eighty years intervening since Kuhlman's publication and in spite of the Chilean developments, the Norwegian synthetic nitrate production, and the difficult commercial situation, the problem has been attacked in a fairly energetic and persistent manner.

At the outbreak of the European war there were two small plants in Europe capable of transforming ammonia into nitric

[1] Chief Technologist, American Cyanamid Co., New York City.

acid, neither of which, however, had shown sufficient commercial possibilities in the five or six years of operation to warrant further expansion. One of the plants was in Germany, but not operating with any regularity, and never had any production capacity to speak of. Not even the military situation had awakened more than a passing interest in the subject in that country. The long anticipated war was to be short. There were at hand enormous stores of munitions that had been accumulated for a dozen years. The country maintained supplies of over half a million tons of Chilean nitrate at all times, and peace was sure to be dictated within forty or fifty days after the declaration of war. Why should the oxidation of ammonia be of pressing importance?

But to return to our history of development. Kuhlman in 1839 passed a mixture of air and ammonia through a glass tube containing spongy platinum. At ordinary temperature no action was observed, but when heated to 300° C. the platinum glowed and with a considerable excess of air in the gas mixture, vapors of nitric and nitrous acids were evolved. At higher temperatures nitrous acid in fairly pure form was obtained. With excess of ammonia the corresponding salts were formed. No quantitative data of any sort were included in the report.

Eighteen years later the work was taken up by Schönbein, who found that if spongy platinum was saturated with aqua ammonia and introduced into air or preferably oxygen gas, slow oxidation to ammonium nitrate took place at ordinary temperatures. Dense platinum caused this oxidation only at higher temperatures. Finely divided nickel and copper also brought about a similar oxidation of ammonia.

Schönbein was probably the first to use dense platinum in the form of wire. He heated a platinum spiral to just below redness and inserted it into a flask which had been rinsed out with a strong ammonia solution. The gas in the flask turned iodide starch solution blue and the flask filled with a white fog. Powdered copper reacted in the same way.

Passing over the next few years briefly: Tessie duMotay in 1871 obtained nitrates by passing a mixture of air and ammonia over lead manganate, permanganate or bichromate heated to 350-400° C. Liebig obtained similar results by using iron oxide.

Weith in 1874 mixed ammonia gas with ozone and hydrogen peroxide for direct oxidation. Twenty years later Siemens and Halske took out a patent on the use of ozone for this purpose. In 1912 Jones and his co-workers of the Semet-Solvay, following somewhat the outline of duMotay, developed the oxidation by the use of alkaline earth plumbates. Nothing commercial has come to date from any of the above schemes and a great many others along the same lines, not enumerated here.

One more name of importance in this early work is that of Warren, who about 1890 passed an ammonia-oxygen or air mixture over platinized asbestos and obtained ammonium nitrate. Although nothing commercial came directly of his work, its publication undoubtedly assisted in causing the cancellation in the German patent office of the celebrated Ostwald patents, which process I shall refer to next.

Twenty-five years ago Wilhelm Ostwald undertook a study of the oxidation of ammonia. He paid much attention to the work of Kuhlman and determined to study the reaction from the scientific standpoint, as nothing along these lines had been done in the fifty years elapsing since Kuhlman's original publication. In his usual careful manner he studied the fundamentals underlying the reaction, and the reading of his exhaustive reports on his investigations would have been of great assistance to many of the recent workers in this field.

Ostwald in general showed the futility of attempting to use spongy platinum as catalytic material, also the necessity of quickly removing the products of the oxidation out of contact with the catalyst. He showed that platinum black was the poorest sort of a catalyzer, yet in spite of this finding, several recent investigators have unsuccessfully attempted to prove otherwise. He noted the activation of dense platinum through operation under special conditions, and particularly the advantage of using such form of catalyst over other types. He further determined the most suitable mixture of air and ammonia for efficient oxidation, and, of the utmost importance, that for such mixture a supply of heat above that directly generated by the reaction was necessary for continuous operation.

The result of this work of Ostwald was the building of two oxidation plants of small capacity, the first of which went into

operation in 1908. The early apparatus installed underwent several modifications, one of the later type being shown here. (Fig. 1.)

The exterior tube was of heat resisting and insulating material and the interior tube of nickel. The catalyzer was built up of narrow strips of platinum foil, alternately crimped and flat, thereby producing a series of short tubes. The ammonia gas-air mixture, in the proportions of approximately 10 percent ammonia, entered the exterior tube at the right, passed to the left in the annular space and out through the red-hot catalyzer and nickel tube. This tube is made in sections to enable cheap replacement, as the sections wear unequally. The hot products of combustion lose their contained heat through the nickel tube to the oncoming raw mixture and so serve to keep the apparatus in operation. The apparatus is started by the use of a gas jet at the catalyzer end, or by burning hydrogen gas around the nickel tube. The catalyzer runs very hot, around 800° C.

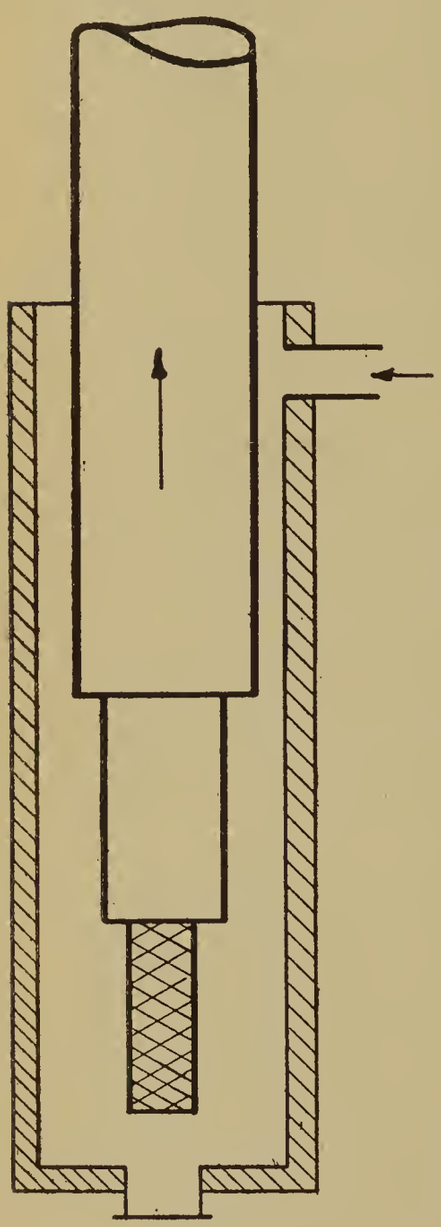


Fig. 1.

A recent form of somewhat similar construction developed by the Barton interests in England since the outbreak of the war, uses a nickel tube about 3 inches (7.5 cm.) diameter and ten feet (3 m.) long. The catalyzer consists of a bundle of spirals of platinum foil hung in the top of the tube. The life of the catalyzer is less than three months. One troy ounce (31 g.) of platinum oxidizes only about 1.5 pounds (680 g.) of ammonia per hour in this apparatus. The reported efficiency is 90-95 percent, but a study of a large installation in France where cyanamid is converted to ammonia and that to nitric acid, shows only 60-70 percent for the whole, and since ammonia production

from cyanamid can be carried out at a nitrogen efficiency of 96-98 percent depending upon the design of the apparatus and the skill of operation, the oxidation efficiency itself must be very low. I have no direct figures on this catalyzer efficiency.

Ostwald himself reported that his plants obtained an oxidation efficiency of 85 percent but my attempts to verify this over long-time operation have been fruitless. I am of the opinion as a result of my investigation that 75 percent would be nearer. He passed his gas over the platinum at a speed of 5 meters per second and left the gas in contact with the catalyzer about 0.004 second, both figures being far outside the region of our later work, and thereby confirm the low efficiency obtained by his process.

Kaiser, a Siemens and Halske chemist, next attacked the problem. He used a platinum gauze for catalyzer and at times directly heated the air, at times activated it by silent discharge, or preferably both, then added the ammonia and passed the mixture through platinum gauze. The literature on the process is so complex as to be really unintelligible. At any rate he reported an efficiency of 150-220 percent, claiming the activated air enabled the attainment of such a high temperature in the gauze catalyzer as to cause part of the nitrogen content of the activated air to unite directly with oxygen. His work was repeated by Kochman, who reported efficiencies of only 105-115 percent. The answer, we later found, lay in the method of analysis used.

In 1908 after the first few cyanamid plants abroad were in operation the European interests backing them placed the world on notice that they were prepared to enter the nitrate field, which up to that time had been controlled by Chilean nitrate and some small arc process product. It looks as if this notice was based on the work of Frank and Caro, which shortly afterwards appeared in the form of patent literature. If so it was very premature as that early process never came into commercial development. The process in question used a mixture of cerium and thorium oxides as catalytic material. Like many other processes of like kind using powdered catalyzers, the process failed because the catalyst sintered and became impermeable after short use. Even as late as 1912 work was still being done on this process, the latest development being a catalyzer consisting of

99 percent ThO_2 and 1 percent CeO_2 and showing 65-70 percent efficiency at best.

The Ostwald patents, after allowance, had been cancelled in Germany and the oxidation process left open to all. Frank and Caro, after the failure of their rare-earth catalyzer, turned to a combination process, passing the ammonia air mixture through a platinum catalyzer and then through a granular mass of cerium and thorium oxides treated with platinum chloride, and resting on a nickel screen above the platinum gauze. The experiment carried out in a small apparatus using a gauze 4 inches (10 cm.) in diameter, showed efficiencies of from 7 to 78 percent with an average of less than 50 percent.

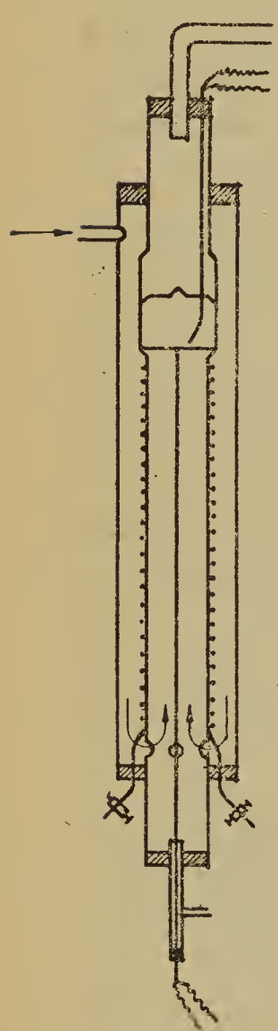


Fig. 2.

In July, 1913, attention of the German cyanamid interests was directed to the use of platinum alone as a catalyzer, and the best features of all previous experiments were assembled in a new type of apparatus. The preheating system of Ostwald, with the direct heating and the gauze of Kaiser, were incorporated in the apparatus shown in Fig. 2. Ammonia could be introduced with the air mixture or at various distances under the gauze. The gauze (1,060 meshes per square centimeter and wire 0.06 mm. diameter) was only 5.5 cm. diameter, not quite all of which was active. From the meager data which came into my possession efficiencies of 41 to 91 percent were obtained, but as ammonia passing the screen unaltered was not regarded as lost, these figures are subject to 23 to 27 percent deduction for unaltered ammonia, making actual oxidation efficiency at a maximum only about 65 percent.

Nothing came of this work, and in September and October, 1913, a new setup of the same apparatus was made. By regulating the ammonia-air mixture to between 1:10 and 1:15 by volume an average efficiency of 85 percent less 6 percent for unaltered ammonia was obtained.

In November a larger diameter apparatus using a gauze 10 cm. diameter was put into service, but the results were not particularly encouraging, rarely exceeding 80 percent efficiency and most of them considerably below that figure.

Experiments were carried out with smooth gauze and with gauze coated with platinum black. The latter showed little or no advantage over the smooth platinum, although slightly less ammonia seemed to pass through unchanged. Various alloys of platinum were tried and it was noted that the presence of iridium had a particularly deleterious effect. Further experiments were carried out using multiple gauzes (that is two and three gauzes superimposed) but in no case were oxidation efficiencies as high as 85 percent obtained. The combination of multiple gauzes and oxides of cerium and thorium gave lower results than the platinum alone.

The effect of mesh of the gauze was also studied and there was little or no difference noted in the behavior of a gauze with 3,000 meshes per square centimeter, as against one with 1,060 meshes. Acetylene and also oxygen was added to the air, but better results were obtained with air alone than with any of the mixtures, though we later, in this country, obtained different results on enriched air.

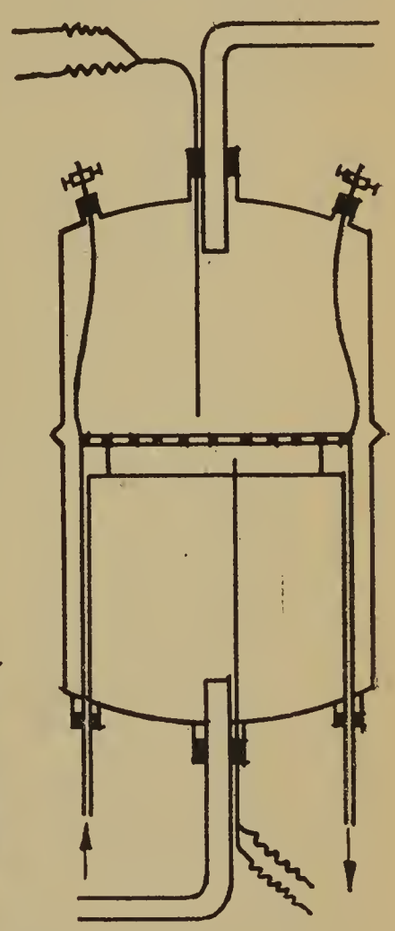


Fig. 3.

At the conclusion of these experiments about the end of January, 1914, this investigation on the oxidation of ammonia had not brought out anything of commercial value. The experiments were critically reviewed and it was concluded that the hot gauze radiated heat to the oncoming ammonia-air mixture and decomposed some of the ammonia. This was emphasized by the necessity of preheating the gases before they arrived at the gauze. It seemed necessary to supply external heat in some way to maintain the reaction and it was, therefore, decided to employ electric energy transformed into heat directly in the gauze itself, for meeting this deficiency of energy, and at the same time to use a cooler in front of and in as close contact

with the gauze as possible, so as to bring the ammonia in the shortest possible time from the ordinary temperature to the reacting temperature which is around 750° C. The apparatus shown in Fig. 3 was constructed to accomplish this result. A long strip of platinum gauze was folded as shown and the intermediate space blocked by mica strips cemented to the gauze. This was placed one millimeter above a circle of heavy nickel gauze which in turn was soldered to a water-cooled copper ring. The apparatus was only a few inches in diameter but it showed very good results, running slightly above 85 percent in conversion efficiency. It was next duplicated on a larger scale in which

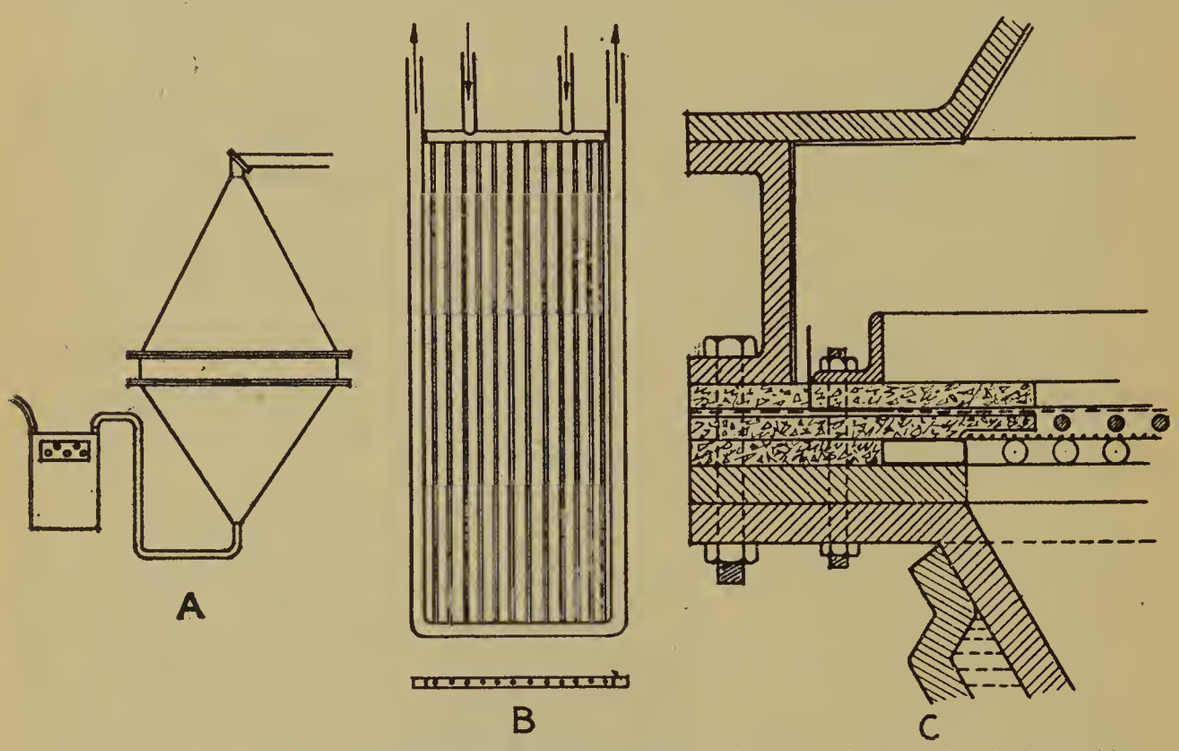


Fig. 4. A, General Assembly; B, Copper Water Cooler; C, Detail of Assembly.

the gauze had an area of 660 square centimeters. This apparatus was put into service, but ran only a few hours when it failed mechanically and it at no time showed an efficiency of oxidation of above 80 percent, although the experiments carried out were the most carefully conducted of any performed.

The general program upon which this work was carried out was entirely different from that of similar investigation work as we do it in America. Strange to say the German experts in charge of it did not seem to devote much time to analysis of the problem from a theoretical or from a physico-chemical standpoint, but instead devoted their attention to the construction of

most elaborate and intricate apparatus, in which the principles of chemical engineering were sacrificed to such external details as convenient existing electrical and mechanical conditions. There had not been a clean-cut analysis of the problem made, as far as I could find up to the time I left Germany in November, 1914.

The construction of a different type of apparatus in which more attention was paid to chemical details next followed. A single sheet of gauze 65 by 25 centimeters was especially made for this work, and very great delay in its manufacture was experienced. This gauze was housed in a metal box lined with aluminum as shown in the detail in Fig. 4. The platinum gauze itself rested upon threads of asbestos which in turn were carried on a nickel gauze. This nickel gauze was soldered to a frame of copper tubes through which water could be circulated. The apparatus was connected electrically to a source of low voltage current through a rheostat, and anhydrous ammonia mixed with air was passed upwards through the cooler and the gauze.

On October 14, 1914, this apparatus was placed in operation. There was recovered in the form of nitrates and nitrites, equivalent nitric acid to the extent of approximately 53 percent of the equivalent ammonia fed to the apparatus. The apparatus failed because the gauze warped away from the cooler in places, and this irregular distance resulted in very non-uniform heating. Conditions did not permit my continuing the study of this experimental work, but at the final conference in which I took part I suggested that the whole apparatus would operate much better if the cooler as installed were abandoned entirely, and the whole catalyzer box turned upside down. This advice was not at the time favorably received, but I believe that later, after my departure from the scene of the experiment, it was adopted and efficiencies of between 80 and 87 percent obtained.

This ended my direct contact with this early foreign experimental work on ammonia oxidation. The impressions I received were not encouraging, as I had not seen an apparatus function for more than a few hours at a time, the highest efficiency of conversion of ammonia to nitric oxide obtained in my presence had been only 53 percent, and no attempt had been made to recover the nitric oxides as nitric acid, but only as nitrate-nitrite.

A record of further German development has been difficult

to compile. On returning to Berlin I there found that the commission in charge of this work had called in a new engineer, one I had not met before, but whom I knew by reputation in nitric acid work. His catalyzer was small, and consisted of a small frame of refractory material about 4 inches (10 cm.) square and ¼ inch (6.3 mm.) thick. Two of these were wrapped with fine platinum wire, very close but not touching, and arranged one over the other so wires crossed at right angles. This gave the gas contact four times with the platinum in a single passage through the apparatus. The frames were arranged in a suitable housing, 24 units to a group. Each unit was supposed to make 230 grams nitric acid per hour, and consumed 4.5 amperes at 90 volts. Mechanically the apparatus was very well designed, but unfortunately it possessed a serious fault in using successive layers of catalyzer not in immediate contact with each other. This arrangement of platinum has never proven satisfactory from an efficient operating standpoint.

On January 1, 1915, the writer is certain that with the exception of the old Ostwald plant, there had been nothing new erected in Germany for the oxidation of ammonia. A year later, however, very considerable installation of ammonia oxidation plants had been made and it seems from the strong hands that the electically heated catalyzer had fallen into, that this made the greatest progress. It is believed that by the end of 1915 some form of an electrically heated catalyzer unit had been developed which was oxidizing at the rate of about 15,000 tons of ammonia per year. It is quite possible that this figure was much underestimated. In 1917 it is believed that the ammonia production in Germany had increased to around 300,000-400,000 tons, part of which was consumed in fertilizer, part in the direct production of ammonium nitrate, and part in the oxidation of ammonia. Just how this division was made, I do not know and have made no serious attempt to learn.

During my stay in Germany I found little or no information on the converting of the nitric oxide obtained from a catalyzer into nitric acid. Much of the apparatus proposed was what would be called small-scale work in this country, consisting as it did of towers of chemical stoneware, tourelles, and such equipment. With such apparatus for condensation, the units must have been

small and inefficient, but on the other hand this type of construction lends itself to ease of assembly and a consequent great saving in time of construction. Also much of the nitric oxide was turned directly into sodium nitrate, that being a comparatively simple and easy problem, and further lending itself to the shipment of nitrate to the various powder plants already provided with the old niter-pot equipment. In fact it is believed that this conversion to nitrate occurred to a very large extent throughout the war.

Toward the end of 1915, when it looked very much as if the trend of events was such that the United States was ultimately to be forced into the war, renewed interest took place in this country in the oxidation of ammonia as a source of nitric acid, and early in 1916 the writer was authorized by the American Cyanamid Company to proceed with the design and construction of an experimental plant for studying the ammonia oxidation problem along the lines of commercializing what had heretofore been a laboratory curiosity. A considerable latitude in the choice of process was permitted, but it was essential that all equipment to be used must be capable of construction in this country. As to raw materials, we were permitted to make use of any form of ammonia on the market, but naturally we were partial to that produced from cyanamid because of a number of apparent advantages which this form of gas was then believed to possess. I am going to briefly sketch here the method of attack of the problem confronting us, as I believe the details will be of interest.

The chemistry of the problem from a quantitative standpoint may be represented by the following equation:

$$4NH_3 + 5O_2 = 4NO + 6H_2O$$

This reaction will only take place in the presence of a suitable catalyzer, of which from a study of all the previous work, platinum appeared to be the best. It is operative between 700 and 800° C., giving its best results at about 750° C.

The best proportions of air and ammonia for most efficient oxidation was reported by Ostwald to be a mixture containing about 10 percent of ammonia. He, however, also expressed the opinion that a very large excess of air was advantageous. The small-scale work abroad seemed to show that a 8 percent mixture

was the most efficient one to use and we subsequently confirmed this in our large-scale work. We, however, later learned that the 8 percent mixture was probably not the most economical one for operation and preferred to work at about a 9 percent concentration.

The combustion of a 10 percent mixture will yield a theoretical temperature of 665-690° C., the difference being due to variations in thermo-chemical constants used in the calculation. Assuming that the catalyzer must operate at around 750° C. and that it will always radiate some heat, it is readily seen that it is absurd to consider attempting to operate the process without the addition of further heat to the catalyzer. With an 8 percent mixture the temperature of combustion is theoretically only 550-580° C.

The reaction—

$$4NH_3 + 3O_2 = 2N_2 + 6H_2O$$

also takes place if the catalyzer is not operating properly, either due to composition or construction, or other causes. The theoretical temperature of the flame corresponding to this reaction when using a 10 percent ammonia mixture is 915° C., using Nernst's data. This is somewhat higher than the actual temperature required on the screen. It is, therefore, seen that it is theoretically possible to operate the catalyzer only if from 25-40 percent of the ammonia carried to the screen is burned to nitrogen, unless one has provided a supply of additional heat to the screen either by regenerative methods or derived from an independent outside source.

In looking over the methods of supplying this required heat to the catalyzer, the regenerative process of Ostwald demanded heating of the reagents to about 300° C., and as at that temperature ammonia is subject to decomposition, the regeneration process did not look technically sound. We also questioned the control of a self-regenerative system in view of our lack of knowledge of the exactness of this control requirement of the process.

The direct-heating scheme used by Kaiser required the combustion of additional fuel and seemed to have no apparent advantages of any sort over the Ostwald system. We dismissed his so-called "activation of air" as of no value.

The only other method that appeared feasible to us was the direct electric heating of the catalyzer itself and since the energy required is not very great, and its cost only a very small fraction of the total operating expenses of the plant, we finally decided to adopt this system.

Our next concern was the type of catalyzer. Both Kaiser and Caro had used the woven gauze or screen type with some degree of success, even though many advantages were claimed for the wire-wound frame. We adopted the screen as our first type of catalyzer and designed two forms, one of 80 mesh (992 mesh per sq. cm.) gauze and 0.0025 inch (0.064 mm.) wire and the other of 100 mesh (1,550 mesh per sq. cm.) with wire of 0.0015 inch (0.038 mm.) diameter. This gauze was to be of purest platinum, free from base metals and containing not over 0.05 percent iridium. Three screens of each type, 13 inches (.33 cm.) wide and 26 inches (66 cm.) long were obtained from the American Platinum Works. The weaving was not particularly uniform and we soon learned that the platinum contained nearly ten times the amount of iridium specified, all of which caused much trouble in the early days of our work.

From the specifications of the gauze we figured out our electrical equipment, which was in itself quite an intricate problem in view of the lack of data on electrical conductivities of such materials as platinum gauzes. The current was received at the laboratory at 440 volts, 3-phase, 60 cycles, and distributed at this voltage for motor circuits. A portion was transferred down to 110 volts for lighting and catalyzer service and to each catalyzer was attached an individual transformer rated 110 v./15.0-17.5-20.0-22.5 volts. In the high tension side were included 3 resistance grids with short circuiting switches and a 25 point rheostat of the same total resistance as a grid. The resistance system was calculated to control the voltage on the low side of the transformer in steps of 0.05 volt over a total range of five volts or two transformer taps. This electrical system was completely revised and both cheapened and simplified in a later installation.

The catalyzer was housed in an aluminum box, rectangular in section. It was stretched in a tray and packed in asbestos board, above which were assembled the sections of the box. We determined to pass the air mixture downwards through the hot cata-

lyzer, believing to insure more uniform distribution and flow by that method. This necessitated some height of tower over the gauze and not knowing just how high this should be, we built the housing in sections, each a foot high, making them interchangeable and providing four sections to each catalyzer unit. An assembly of the tower is shown in Fig. 5. Between the upper sections we placed an aluminum gauze of fine mesh and heavy wire to assist gas distribution, and at the same time carry heat to the side walls where it was dissipated. The four foot high tower seemed to be just right and has not been changed since the original design, except to eliminate the sectional feature in the later and more permanent installations. The catalyzer trays were all interchangeable and enabled us to repair and change gauzes with very little interruption to operation. After getting into operation we did not find it necessary to complicate the apparatus with a cooler, for while we believe in the use of a proper cooler we could not design a satisfactory one.

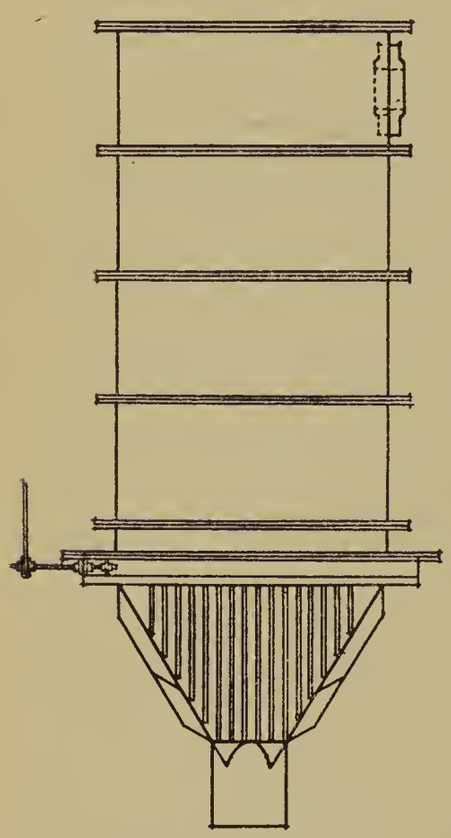


Fig. 5.

Besides the oxidation equipment, the plant was equipped with an autoclave and ammonia apparatus for the production of ammonia from cyanamid, gas holders, and most complete measuring and indicating apparatus. Also oxidation, cooling and absorption towers for the recovery of nitric acid. Nothing was left to chance and for our efforts spent on the elaborate design we were amply repaid, for with the exception of the first platinum screens and their terminals, every piece of apparatus installed performed to the highest expected degree.

The first screens being delivered during the erection of the plant, we took one of them to the cyanamid plant in Canada, and tried it out there on a small scale, absorbing the products of the reaction in caustic soda. We had learned from Ostwald that activation of the platinum was necessary but did not then know what

it meant or how to obtain it. After running some two weeks with results showing up better each day, we finally, on April 24, 1916, obtained an efficiency of oxidation of 91 per cent. In this series of experiments the highest figure obtained was 92.4 percent, at a rate of 2 pounds NH_3 per hour per square foot (9.76 kg. per 1 sq. m.) of screen. The screen used weighed about 1.5 troy ounces per square foot (33 mg. per 1 sq. cm.).

While at Niagara the use of enriched air was tried in the mixture and while it gave very good chemical results, the problem of control was so great that we did nothing to put its use into practice. By its use outside energy can be dispensed with, at least to a very large extent, but the problem of distributing a self-combustible gas over a large area of catalyzer, will require more experimentation before it is worked out to a certain war-time surety.

Toward the end of July, 1916, the small demonstration plant at the Ammo-Phos Works of the American Cyanamid Company at Warners, New Jersey, was put into operation and it fulfilled our expectations in every way. The very light weight screens first provided proved themselves structurally too weak for continuous heavy service and we abandoned their use. As a matter of fact we later increased the weight of our regular screens about 50 percent purely for structural reasons. We also found that by passing the gas downward through the converter, only a trace of ammonia passed the screen in its original form. It was either burned or decomposed. We also learned that the capacity of a screen could be increased from the two pounds of ammonia per square foot per hour as obtained in our preliminary work, up to seven pounds of ammonia per square foot (34 kg. per sq. m.) of screen per hour, without greatly affecting the efficiency, and under special test conditions we actually succeeded in operating up to ten pounds of ammonia per square foot (49 kg. per sq. m.) of screen per hour. As a steady operating proposition, however, this latter rate would result in a lowering of the efficiency of oxidation two or three percent, and we actually figure for most economical operation, with the high cost ammonia, 5.5 to 6 pounds per square foot (26.8 to 29 kg. per sq. m.) of screen per hour.

The highest efficiency attained in all of our development work was 96 percent, but this was under particularly careful operating

conditions at a rate of five pounds of ammonia per square foot (24.4 kg. per sq. m.) of screen per hour.

Operations of the plant were comparatively simple after conditions were standardized. Ammonia gas and air were received at constant pressure and fed through meters and valves for mixing. The control of the mixture was a very simple matter and gave no trouble whatever. The screen attendant would turn on the current to a catalyzer and then feed in the gas mixture at a predetermined rate indicated by an orifice gauge on each catalyzer tower. He would then regulate the current so as to obtain the proper temperature, at first measured by pyrometer and later by eye. Two men per shift have operated one hundred catalyzer towers over a considerable period of time, the production of nitric acid during this period averaging one-half ton of 100 percent equivalent acid per catalyzer per day.

After eliminating the mechanical defects of the early screens we adopted a heavier type, weighing slightly less than 2½ ounces per square foot (55 m.g. per 1 sq. cm.) and we have had some of these screens in service two years continuously on cyanamid-ammonia gas as obtained direct from the autoclave plants and without any purification whatever. These screens seem to be in just as good catalytic condition today as when first activated.

The life of the platinum in this process has been the subject of much discussion. Our experience has been that a platinum catalyzer in the ordinary operation of a plant loses between 3 and 5 percent of its weight per year. Including accidents it looks as if a screen should give good average service for about one year, at the end of which time it will have lost less than 5 percent of its weight. It might run longer before being melted, redrawn and rewoven, but we do not have sufficient large-scale experience at our command to give the final answer. In the early days of our experience we had many mechanical accidents and we thought this deterioration of platinum would be considerably greater than the above figures indicate, but on the basis of our experience with the large Government plant, where a very careful check of all platinum was kept, we feel certain at least of the above figures for deterioration and period of renewal. With more skilled operators, and higher development of equipment, under conditions more favorable to manufacture than during the war, everything

points to a very long life of catalyzer when used on cyanamid-ammonia. Coke-oven ammonia seems to present a different aspect. Several times during the past year there has appeared in print some rather serious criticisms of ammonia derived from cyanamid, particularly in reference to its content of impurities harmful to the platinum catalyzer, as well as to its general application to oxidation. The first published criticism laid the blame on acetylene. Cyanamid-ammonia as obtained directly from the autoclaves, which are producing ammonia gas for fertilizer manufacture, does contain a very small fraction of one percent of acetylene gas, but we never found any difficulty in oxidizing such acetylene-ammonia mixtures, provided the acetylene content did not get so high that when mixed with air the resulting gas was self-explosive when conducted to a hot catalyzer. The authors of this first criticism, after rushing into print, later found that their conclusions were entirely in error and that it was not acetylene that was causing the trouble, the true cause then became phosphine. They again rushed into print with a second paper showing the evil influence of this phosphine. Autoclave gas undoubtedly does contain traces of phosphine at times, but the quantity is so small that present-day quantitative methods are unable to detect it with any degree of precision. Having never used any other but such autoclave gas in our laboratory and having no trouble whatever owing to poisoning of the platinum catalyzer, even with screens in use two years, we did not pay very much attention to the matter at first. However, as the criticism of cyanamid-ammonia became a matter of public discussion we thoroughly investigated the whole subject and learned to our surprise that carefully purified autoclave gas, in which it was freed from acetylene, phosphine and other possible impurities, showed consistently about 10 percent less efficiency on the same screen as when we used the raw gas. This was a most curious state of affairs and after we had firmly established this surprising difference under the most careful control possible, we learned that the platinum catalyzer seems to activate itself slowly to each particular and peculiar gas to which it is subjected during activation and that a catalyzer which had been activated on autoclave gas will not work efficiently on purified autoclave gas or on coke-oven ammonia, but requires a wholly new activation. Left to itself

the period of adjustment from one form of activation to another is about ten days to two weeks, whereas our critics were subjecting their catalyzer to ten different kinds of gas on the same day and naturally were drawing some very erroneous conclusions. We found a gauze activated on coke-oven ammonia could not be used efficiently an cyanamid-ammonia and that one activated on cyanamid-ammonia could not be used efficiently on coke-oven ammonia, on immediate turn-over, and that it took at least ten days of steady operation on the second variety of gas before the efficiency arose to where it was on the kind of gas the catalyzer was activated on. We then found on examining the catalyzer closely that there were structural differences in the form which the platinum took when activated with different varieties of ammonia gas.

I have here a slide containing a bit of platinum gauze as received from the weaver and also a piece of gauze which has been in service for some time; the difference on direct vision is quite striking. I have here also a piece of the full-sized catalyzer which has been in service for a long time showing this effect of activation. Some of our most insistent critics, when thoroughly educated by us to these facts have later asked to be given ammonia gas high in acetylene and phosphine in the hopes that through its use their own impossible apparatus could be made at least to operate to somewhere near the efficiency of our own type. It has been most harmful to the progress of the development of the oxidation of ammonia in this country that such individuals as were responsible for the publication referred to above, ever interested themselves in the subject. It will take much honest endeavor to discredit the false information spread broadcast.

I can only add here that the cyanamid process was installed in the Government Nitrate Plant at Muscle Shoals, Alabama, the whole equipment there embracing 700 catalyzers capable of easily producing with inexperienced attention, about 250 tons of equivalent 100 percent nitric acid per day, and which could be pushed under emergencies and demands, to 50 percent greater output or 350-375 tons per day, without an extreme sacrifice in efficiency. The autoclave plant transformed cyanamid into ammonia at an efficiency of 97 percent, as measured over an extended test under

independent Government control. In this test the plant was operated by men, part of whom had been trained in the American Cyanamid Company's Experimental Plant, but the chemical control was entirely exercised by Government chemists. The catalyzer efficiency for all catalyzers over this same period of time was 91.5 percent. The absorption efficiency to the production of nitric acid was 98 percent. This is equivalent to a total efficiency from cyanamid to nitric acid of 87 percent, and of ammonia gas to nitric acid of practically 90 percent. With more experienced operators, undoubtedly these figures would have been raised in the next six months, several percent. The above figures are certainly a direct contradiction of a statement of some of our critics that cyanamid-ammonia cannot be oxidized at any high degree of efficiency because of its impurities. As a further matter of interest the power requirement of the process as installed, averaged less than one-seventh of a kilowatt-hour per pound (less than one-third per 1 kg.) of nitric acid made on the catalyzer.

The gases leave the catalyzer at practically redness, although attempts are made to radiate as much heat as possible. They enter flues under the floor and are carried to coolers where their temperature is brought down to as near normal as possible. They then pass into oxidation chambers where a minute or two of rest converts practically all of the nitric oxide to nitrogen dioxide. This passes into absorption towers where oxidation to nitric acid takes place and the acid is absorbed in water. Due to high concentration of nitric oxide gases, the efficiency of recovery in our absorption system is very high. In the Government plant it averaged above 98 percent and frequently ran for long periods of time at 99 percent. All absorption was done in water and no soda towers were used in the system whatever. The strength of acid obtained could be run as high as 50 percent or even more, but as the material was to be used for making ammonium nitrate, no attempts at obtaining extreme concentrations were made. Since this nitric acid is extremely pure, circulation through aluminum pipes is entirely feasible and possible, and its handling with this facility at our disposal was a simple problem.

As to the cyanamid oxidation process, it does not lend itself as readily to very small-scale installations as the old niter pot,

because of the cost of installation, but where ammonia gas is available and it is not necessary to provide separate ammonia equipment, even in this case it will find some very interesting uses. For instance at the Ammo-Phos Works of the American Cyanamid Company, a catalyzer operating on ammonia gas is supplying nitrogen oxides direct into a sulphuric acid chamber plant, with very great success and at very good economy over the old potting process. Not only are the nitrogen oxides much cheaper than when made from nitrate of soda but the consumption is very much less and the control of the chamber plant very much simplified.

With ammonia gas of suitable quality obtainable at no higher unit price than equivalent nitrogen in nitrate of soda, and on a reasonably large operating scale, the oxidation process will make concentrated nitric acid at at least as low a cost as the niter pot and on a large scale at a lower cost. The cost of the plant is somewhat more than for a similar sized pot plant, but the consumption of raw materials and of labor, and the ease of manipulation, all greatly favor the ammonia oxidation plant, and it is quite likely that where the size of the installation is sufficiently great, it will prove a very serious competitor of the old nitrate of soda process.

On a fair-sized plant, the operation, concentration, overhead, taxes, interest, and depreciation will amount to approximately one-third of the cost of concentrated nitric acid, but this can only be accepted as a most general figure, because each plant and location must be studied individually as these factors exert a very considerable influence on the total cost of the product. It must further be remembered that not all varieties of ammonia have shown themselves capable of oxidation. The Muscle Shoals Plant furnished some rather interesting data on the cost of nitric acid and ammonium nitrate, the factory cost on the latter being less than one-half of the Government fixed price, from such figures as were obtained during the short period of partial operation of that plant. The cyanamid oxidation process has been fully protected by patents and its development and installation is still wholly in the hands of the American Cyanamid Company.

In conclusion the early prophecy of Kuhlman is about to be fulfilled and large consumers of nitric acid will do well to look

closely to the oxidation of ammonia in planning their future extensions. As a wartime emergency the development in all the warring countries has given us a direct answer to its practicability. Without it Germany would not have lasted a full year of modern warfare, and the Allies themselves were forced to come to it, even with Chile open to them. When coupled with the fixation processes its saving in transportation, raw materials, sulphuric acid, and its adaptability to large-scale units place it far ahead of the older nitric acid process, and for certain special applications it has no competitor. It was a long time coming to the front, but I believe it has now reached the point where future extensions and improvements will be rapid and I anticipate it will soon be producing its better product alongside its older competitor on a fair-sized scale.

DISCUSSION.

W. S. LANDIS: I have here two of the catalyzer screens used by the students in the Warner plant. They are perfectly active catalytically, but are unfortunately badly torn at the ends and full of holes, so we would no longer consider their use. I suppose this screen is good for 85 percent efficiency in its present condition.

J. W. RICHARDS[1]: What would be the difference between the temperature to which the mixture was heated and the temperature to which the platinum was heated by the heating current? Have you any temperature determinations?

W. S. LANDIS: Unfortunately in most of this work we sacrificed scientific information to production, and one of the things that we sadly lack is an accurate measurement of temperatures. That is due to the fact that the ordinary pyrometer and the ordinary apparatus which we commercial men possess, does not apply in any of this work. It is impossible to touch the screen with a thermo-couple, because it cools the gauze by mere contact. The windows or apertures are mica and soon cloud up and seriously affect the optical pyrometer. We never even took the time to calibrate these windows. Ordinarily the amount of energy used

[1] Prof. of Metallurgy, Lehigh University.

in any of these catalyzers does not run anything near in proportion to the amount of air mixture you are passing over the catalyzer. Whether you are running at a thousand or five thousand feet an hour, it does not mean that you take five times as much energy for five thousand as for one thousand. If running at a thousand feet an hour, it would require five or five and a half kw., and if running at six thousand feet an hour, it might consume six and a half; ordinarily the current put into that screen brings it to just about glowing in the dark, or possibly a little brighter than just glowing in the dark, and that is sufficient to operate at almost any capacity up to extreme ranges, where a few percent more is required. Apparently the function of the current is to take care of radiation and to do very little pre-heating.

J. W. Richards: But actually the screen must be at a considerably higher temperature than the temperature of the mixture?

W. S. Landis: I doubt that. If, in operating one of these screens we pass ammonia gas downwards, the only thing we have is a more or less visible flame underneath the catalyzer. Whether that flame is at a higher or lower temperature than the screen, I do not know. I am inclined to believe there is very little difference in temperature between them. It is something the scientific man, with plenty of equipment in his hands, will have to study.

J. W. Richards: I asked the question because of the loss of weight of the platinum, and I could not think of its being due to anything else but vaporization. If the temperature were 300 degrees hotter than the working temperature of the mixture, the loss would probably be accounted for by vaporization of the platinum.

W. S. Landis: I think that loss is due to the mechanical abrasion of the gas passing over the crystalline structure of the wire. These crystals, under a high power microscope, take on various forms. Some of them appear as a mushroom and others as a tree. We believe that the loss is due to the mechanical wear of these crystals due to the gas passing over them.

Colin G. Fink[2]: The point that Dr. Richards brings up about the vaporization of platinum sounds more plausible because those crystals on the wire could not be formed unless the platinum did

[2] Head of Labs., Chile Exploration Co., New York City.

vaporize either as platinum or as a platinum compound. I do not believe that you could get the platinum crystals unless you had some platinum compound present which decomposed rapidly and formed platinum crystals. Part of these platinum compounds might be carried away with the gases and thus account for part of the loss. (See Chemical Abstracts, 1907, pp. 2849-50.)

C. A. DOREMUS[3]: Is there any microscopic change visible when you activate from one substance to another, from one kind of ammonia to another kind of ammonia?

W. S. LANDIS: Not only is there a microscopic change, but you can actually see it with the naked eye. Some platinum screens, when activated, take on a very dull matt surface, particularly when they have been activated by coke oven ammonia. Some others take on a brilliant crystalline appearance. I have not made a thorough study of that activation with all the physical conditions surrounding it, but I think it is worth while for somebody to take that up.

C. A. DOREMUS: Suppose you have been using this process with the cyanamid screen and change to coke oven gas, will that change its appearance?

W. S. LANDIS: It will. I have not done it in that way; I have changed from coke oven gas over to cyanamid; whether it would go back, I cannot exactly say, as such an experiment would take about four weeks of steady operation; that is two weeks per change.

C. A. DOREMUS: I might incidentally state, gentlemen, that this comes close to some of the discussions Dr. Bone had in regard to surface combustion and the change of surface due to the use of combustible gases which he showed in this hall, and photographs of screens that were used in surface combustion.

L. E. SAUNDERS[4]: It seems to me that the question raised by Dr. Fink must have had some animus behind it, and if he thinks the results were due to the vaporization of platinum, he has some reason for it and I would like to hear from him as to what his reasons are.

[3] Chemical Engineer, New York City.
[4] Mgr. Abrasive Plants, Norton Co., Niagara Falls, N. Y.

COLIN G. FINK: It would take too much time to explain that to Mr. Saunders, but suffice it to say that under similar conditions when a platinum wire-wound alundum tube is packed in white alundum powder, with which I believe Mr. Saunders is familiar, and heated in an atmosphere of hydrogen, we find upon examining the packing after a few hours' operation that platinum crystals are deposited on the alundum grains half a centimeter away from the platinum wire.

L. E. SAUNDERS: What temperatures are you speaking of?

COLIN G. FINK: You do not need to go much above red heat. It depends upon the gases you are using. What we are getting here is some volatile platinum compound which is readily decomposed and during the process of decomposition it is precipitated in a definite crystalline modification upon the alundum grains.

W. H. FULWEILER[5]: What is the poison that has to be avoided in this process?

W. S. LANDIS: I have not found any poisons that have to be definitely avoided in the process. I think that platinum will handle almost anything you want to feed to it, within reason. Acetylene has been blamed as a very active poison; my experience is that it is not until you get up to a concentration where it actually deposits lampblack on the platinum. Phosphine has been claimed to be a particular poison. I have been working with gases that contained phosphine, how much I do not know, but I have not found it to seriously affect the screen. Sulphur seems to be a particularly vicious material, not so much on the screen as on the rest of the apparatus. For instance, one great difficulty with coke oven ammonia has been sulphur. It oxidizes to sulphuric acid, mixes with the nitric acid and your aluminum piping and absorption towers corrode very rapidly. There are certain forms of sulphur that apparently affect the screen to some slight extent. They are probably organic forms; what they are and how much, I cannot tell. We have taken cyanamid-ammonia direct from autoclaves and oxidized it without purification in any way. We have had poor results from coke-oven ammonia.

W. H. FULWEILER: What do you mean by coke-oven ammonia?

[5] Chief Chemist, United Gas Improvement Co., Philadelphia, Pa.

W. S. LANDIS: Ammonia derived from coal distillation in its purer forms.

W. H. FULWEILER: In recovering ammonia from coke ovens or retorts, the ammonia is absorbed either with water forming a weak ammonia liquor which is afterwards concentrated to about 16 percent ammonia content or the ammonia may be absorbed directly by sulphuric acid forming ammonium sulphate. The concentrated ammonia liquor may be again treated to form anhydrous ammonia. This, on boiling, delivers the free ammonia again, or the ammonium sulphate may be decomposed with lime. The purity of the ammonia gas will naturally vary to a very considerable extent, depending upon whether it is derived from a concentrated liquor, anhydrous ammonia or ammonium sulphate, and I should like to know which method was used in making the ammonia gas that is referred to as coke-oven ammonia.

W. S. LANDIS: In all the use which we have made of coke-oven ammonia, it has generally been in the form of anhydrous ammonia, which is the most satisfactory to work with, because you can put your cylinder on the scale and weigh it, or in the form of a highly purified liquor. I am speaking about coke-oven ammonia to distinguish from cyanamid ammonia. I have never used the crude liquor.

HAROLD HIBBERT[6]: I wish to ask Dr. Landis the percentage of sulphur in the ammonia?

W. S. LANDIS: I could not tell you. It runs a few hundredths of a percent in most anhydrous ammonia. In aqua-ammonia it is sometimes below a hundredth of a percent, but not very much. We have not used crude liquor at all in our process. Most of our work has been done with cyanamid ammonia taken directly from the autoclaves producing it and without purification.

PAUL POETSCHKE[7]: I would like to add a statement in regard to this volatilization of platinum. I do not believe that crystallization is necessarily due to volatilization, because every chemist is familiar with the fact that in the presence of a reducing agent platinum will crystallize, and I doubt whether crystallization is necessarily connected with volatilization.

[6] Research Chemist, New York City.
[7] Director Dept. of Chem., The Pease Labs., New York City.

Presented as part of a Symposium on "Released Electrochemical Information" at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

SILICON TETRACHLORIDE.

By Otis Hutchins.[1]

In the fall of 1917 the Ordnance Department requested three electrochemical plants at Niagara Falls to develop processes for the production of silicon tetrachloride, and to put themselves in position to manufacture the material on a large tonnage basis. One of these concerns—The Carborundum Company—developed the process to a commercial stage, although it did not participate in the government contract except to supply silicon and silicon carbide for the further production of the tetrachloride. The process it developed, however, followed the general lines of the process used in the large-scale operations, and it is believed that a description of some of the experimental work will be of interest.

There is no evidence that silicon tetrachloride was other than a laboratory product before the war. However, there does appear to have been a considerable number of investigations made on the properties and reactions of this compound.

Silicon tetrachloride, when pure, is a colorless, volatile liquid, containing 16.65 percent silicon and 83.35 percent chlorine. Its composition and vapor density correspond to the formula $SiCl_4$. The properties of this material, with references, are given below:

Specific gravity at 0° C.: 1.52.
(Thorpe: J. Chem. Soc. (1880) **37**, 327.)

Melting Point: —89° C.
(Abegg: Handbuch der anorg. Chem.)

Vapor Pressure:
(J. Meyer & Becker: Z. anorg. Chem. (1905) **43**, 209.)

0° C.	116.0
5	133.5
10	150.0
15	175.0
20	216.0
25	261.5
56.9	760.0

[1] Metallurgical Engineer, The Carborundum Co., Niagara Falls.

Vapor Pressure:
(Regnault, Mem. de L'Academie (1862) **26**, 334.)

—20° C.	26.49	mm. Hg.
—10	46.46	" "
0	78.02	" "
10	125.90	" "
20	195.86	" "
30	294.49	" "
40	429.08	" "
50	607.46	" "
60	837.23	" "

Boiling Point: 56.9° C.

Heat of Formation: 128,800.
(Richards, Metallurgical Calculations.)

Coef. of Expansion:

$$V = V_0 + 0.001330946T + 0.0_6\,280978T^2 + 0.0_6\,215657T^3$$

(Thorpe: J. Chem. Soc. (1880) **37**, 327.)

Specific Heat: 13° C. — 50° C. 0.1995.
(Ogier: Compt. rend. (1879) **88**, 972.)

Molecular Heat of Evaporation: 6.36 Cal. per gram molecule (37 cal/g.)
(Ogier: Compt. rend. (1879) **88**, 971.)

Index of Refraction at 20° C.:

For line n α of Hydrogen Spectrum.....1.4119
For line n β of Hydrogen Spectrum.....1.4200
For line n γ of Hydrogen Spectrum.....1.4244

(Haagen: Pogg. Ann. (1867) **131**, 117.)

Dissociation Point: above 1600° C.
(Berzelius, Pogg. Ann. (1824) **1**, 219.)

Critical Temperature: 230° C.
(Mendeleeff, Ann. (1861) **119**, 11.)

Heat of reaction with water, per gram-molecule: 69,000 cals.
(Berthelot, Ann. Chim, Phys. (1878) (V) **15**, 213.)

Numerous methods of preparation on a laboratory scale have been used or suggested—

1. Passing chlorine over silicon at red heat.
(Abegg: Handbuch der anorg. Chem.)

2. Passing chlorine over a mixture of silica and carbon at red heat.
(Abegg: Handbuch der anorg. Chem.)

3. Passing chlorine over a mixture of SiO_2 and Mg. at 300-310° C.
(Gatterman & Weinig Ber. (1894) **27**, 1943.)

4. Passing chlorine over copper silicon at red heat.
(Ber. (1894) **27**, 1943.)

5. Passing chlorine over ferrosilicon at red heat.
(Ber. (1894) **27**, 1943.)

6. Passing BCl_3 over SiO_2 heated to dull redness.
(Troost & Hautefeuille: Ann Chim. Phys. [5] **7**, 476.)

7. Passing HCl and vapor of naphthalene over heated silicon.
(Favre: Compt. rend. **107**, 339.)

A study of the composition of silicon tetrachloride would indicate that there is little to be gained in trying to find a cheap source of silicon for the manufacture of this compound, if, in utilizing a given silicon compound, the process is complicated to any considerable degree. On the other hand, a cheap supply of chlorine would be highly desirable.

A full consideration of the question indicated that an economical process must be based either on the reaction of chlorine on silicon or the reaction of chlorine on silicon carbide, or a combination of the two. The reaction with pure silicon, $Si + 2Cl_2 = SiCl_4 + 128{,}800$ cal., which takes place energetically at 1,000° C., is highly exothermic, and when once started on a large scale proceeds rapidly without additional heat until the silicon is entirely consumed. This process can be made continuous, and it is in all probability the simplest process for the manufacture of silicon tetrachloride.

The research department of The Carborundum Company had, at various times, made small quantities of silicon tetrachloride from chlorine and silicon and from chlorine and silicon carbide, and knew something of the merits of the two reactions. However, owing to the fact that a chlorine manufacturer was investigating the silicon reaction, The Carborundum Company turned to the silicon carbide reaction, and, in September, 1917, began experiments on the commercial production of silicon tetrachloride, according to the following reaction—

$$SiC + 2Cl_2 = SiCl_4 + C + 102{,}280 \text{ cal.}$$

This reaction has certain disadvantages as compared with the silicon reaction. The temperature of reaction is higher, a carbon residue remains and the heat evolved by the reaction is not suffi-

cient to maintain the temperature of the reacting mass unless special precautions are taken to conserve heat by insulation and by working with large units.

Our first experiments consisted in placing silicon carbide powder in a quartz tube 1 inch (2.5 cm.) inside diameter, which was heated by ni-chrome wire to 900-1,000° C. Chlorine gas obtained from liquid chlorine was passed through sulphuric acid to remove any moisture (this was later found to be unnecessary), and passed over the silicon carbide and then through a water-cooled condenser. Temperature measurements indicated that this reaction started at about 1,000° C., but proceeded extremely slowly at this temperature.

We next tried passing chlorine over silicon carbide, placed in a carbon tube which was heated by passage of an electric current. This experiment was a failure, owing to the porosity of the carbon tube.

A quartz tube was then used embedded in a granular graphite resistor which was heated by the passage of an electric current. In these experiments the condenser consisted of an iron pipe surrounded by ice water. Temperatures up to 1,250° C. were obtained, at which point the reaction between chlorine and silicon carbide proceeded energetically. Two hundred and twenty-five grams of silicon tetrachloride was made in this apparatus before the quartz tube failed because of its recrystallization.

These experiments brought to our notice several difficulties which would be encountered in the manufacture of this material. First, there was the very troublesome reaction between silicon tetrachloride and water, which may be represented as follows:

$$SiCl_4 + 2H_2O = SiO_2 + 4HCl.$$

The hydrolysis takes place instantly when any moisture comes in contact with silicon tetrachloride, with the resulting deposition of silica and stoppage of the pipes. This reaction necessitates keeping the entire generating system and containers free from moisture, and demands the use of perfectly dry chlorine. Another difficulty encountered was the formation of iron and aluminum chlorides. This was especially troublesome when using silicon metal as a source of silicon, as this material contained, in addition to 90 percent silicon, 7 percent iron and 2 percent

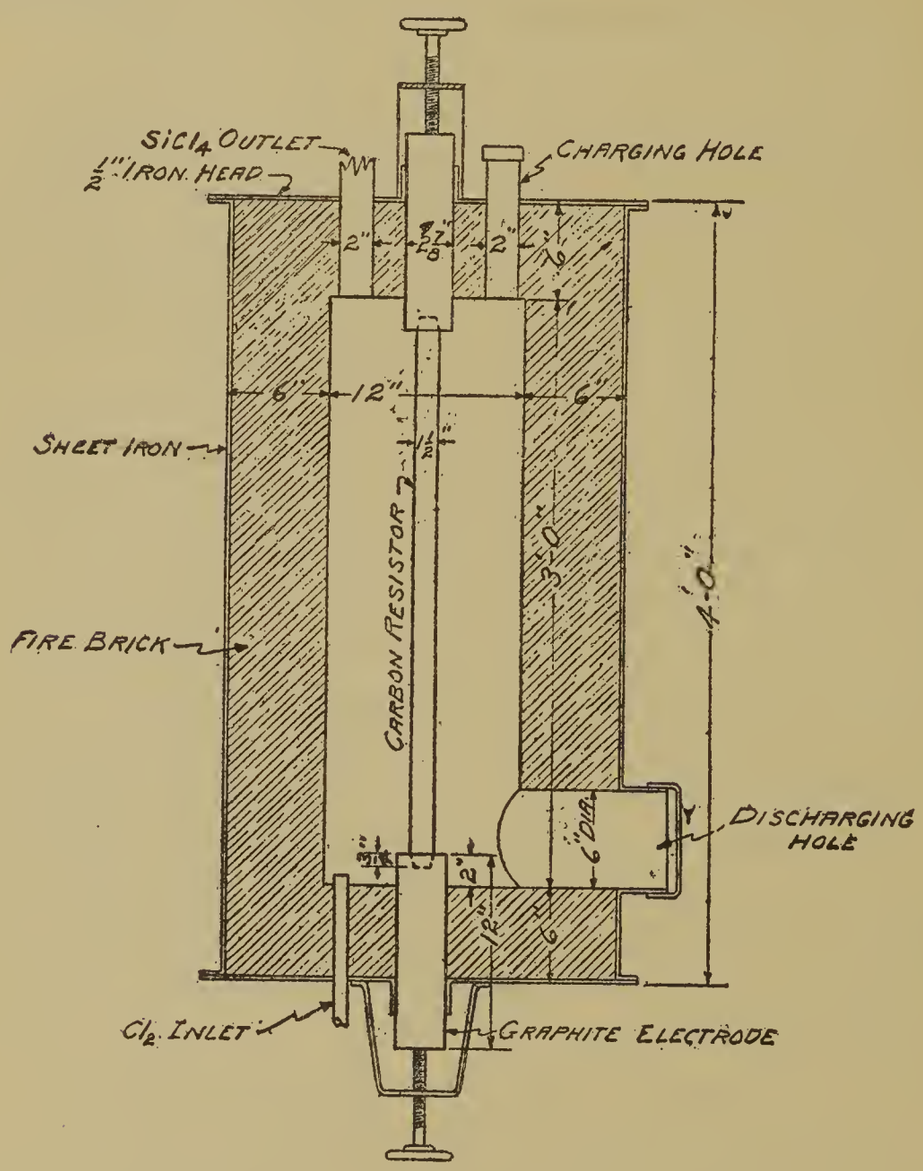
aluminum. The greater portion of the iron and aluminum chlorides was deposited in the piping as soon as the silicon tetrachloride gas left the highly heated part of the system. Silicon tetrachloride dissolves a small percentage of these chlorides as well as some chlorine. Due to this fact, commercial silicon tetrachloride is yellowish in color; redistilling over mercury and collecting the distillate between 55° and 62° C. produces a colorless product.

It was evident that to manufacture this material on a large scale from silicon carbide it would be necessary to have a gas-tight chamber which could be heated to 1,200° C., means of charging the silicon carbide and discharging the carbon residue, and to have a trap of suitable design, connected to the outlet of the chamber, in which could be deposited the bulk of the iron and aluminum chlorides.

Our first selection of a chamber was a zinc retort built into a brick furnace and arranged so that it could be heated with an oil flame. This apparatus was a complete failure, as it was found impossible to make gas-tight joints and the zinc retort invariably cracked before any silicon tetrachloride was obtained. We next turned to electric heating, and designed and built a successful furnace which had a power rating of 15-20 K.W. and which had a capacity of 20-30 lb. (9-14 kg.) of silicon tetrachloride per hour. This furnace, as shown in the following diagram, consisted of a sheet-iron shell, flanged at ends, 24 inches (60 cm.) in diameter and 48 inches (120 cm.) high. The bottom and top of the furnace consisted of cast-iron plates which were bolted to the flanges of the shell.

A layer of insulating material was placed between the shell and the heads, and all bolts were insulated so that there was no electrical connection between the heads. In each head was a 3 in. (7.5 cm.) hole, around which was welded a 3 in. (7.5 cm.) pipe-coupling to act as a guide for the graphite electrode. The graphite electrodes were 2⅞ in. (7.2 cm.) diameter by 12 in. (30 cm.) long, and had a coned recess into which fitted the carbon heater. This carbon resistor was 1⅝ in. (4.1 cm.) diameter by 36 in. (90 cm.) long, with ends tapered to fit the recesses in the graphite electrodes. Pressure was brought to bear on the electrodes, forcing them in close contact with the resistor, by

means of suitable clamps at top and bottom of the furnace. The space between the electrode and the pipe-coupling was filled with "J. M. Vitrex" cement. This joint was not entirely free from leakage, and could be much improved upon. Current was introduced into the graphite electrodes by means of water-cooled copper terminals.



SILICON TETRACHLORIDE FURNACE

FIG. 1.

The top head of the furnace had welded to it a 2 in. (5 cm.) exit pipe for silicon tetrachloride gases, also a 2 in. (5 cm.) pipe for introduction of fresh charge. The lower head had welded to it a 1 in. (2.5 cm.) chlorine inlet pipe. A six-inch (15 cm.) hole was cut in the shell casing 6 in. (15 cm.) above the bottom of the

furnace. A 6 in. (15 cm.) pipe-coupling was welded around this hole, and this opening into the furnace was used for discharge of carbon residue. Both the discharging and the charging holes were closed by means of caps. The interior of the shell was lined with firebrick carefully cemented together, the size of the reacting chamber being 12 in. (30 cm.) diameter and 36 in. (90 cm.) high.

A trap was connected to the gas exit pipe of the furnace. This trap was made of sheet iron, had a removable cover and removable caps on top of the cover to be used for cleaning the inlet and exit pipes in case of stoppage. The inside dimensions of this trap were 30 by 6 by 6 inches (75 by 15 by 15 cm.).

Immediately attached to the trap was the condenser. This consisted of a sheet-iron box 36 in. (90 cm.) high, 12 by 12 in. (30 by 30 cm.) in cross section, with removable cover for cleaning, and contained inside 27 ft. (8.1 m.) of ½ in. (1.25 cm.) iron pipe, through which cold water passed. The silicon tetrachloride gas was introduced into the bottom of this condenser by means of a 2 in. (5 cm.) pipe. A 1 in. (2.5 cm.) exit pipe at the top allowed uncondensed gas to escape. There was an outlet in the bottom of the condenser to draw off the condensed liquid. In most of the runs on this system a secondary condenser was used to determine how much silicon tetrachloride could have been condensed had the temperature been lower. This condenser consisted simply of a U pipe surrounded with ice and salt and maintained at a temperature of —10° C. The assembly of the apparatus is shown in accompanying diagram, Fig. 2.

In making a run on this furnace, about 80 lb. (36 kg.) of silicon carbide was introduced into the reaction chamber. Electric power was supplied to the resistor at about 12.5 volts, the power consumed being 15 K.W. At the end of two hours, the furnace was sufficiently hot to introduce the chlorine. Liquid chlorine was used, two or more chlorine tanks being connected in parallel to the furnace. The pressure of the chlorine entering the furnace varied between 30 and 10 cm. of water. Great difficulty was encountered in keeping this pressure up, due to cooling of the tanks caused by evaporation of the liquid chlorine. Shortly after chlorine gas was admitted to the furnace, silicon tetrachloride began running out of the condenser. It was generally necessary

to cut off the electric power as soon as the reaction got well under way, due to heat generated within the furnace by the reaction.

The maximum production of $SiCl_4$ was 30 lb. (13.6 kg.) per hour, the average being 20 lb. (9 kg.) per hour. This output could have been increased had chlorine at a greater pressure been available. The temperature of the gases entering the first condenser was about 200° C. The usual temperature of the uncondensed gas, issuing from this condenser, was 18° C. The

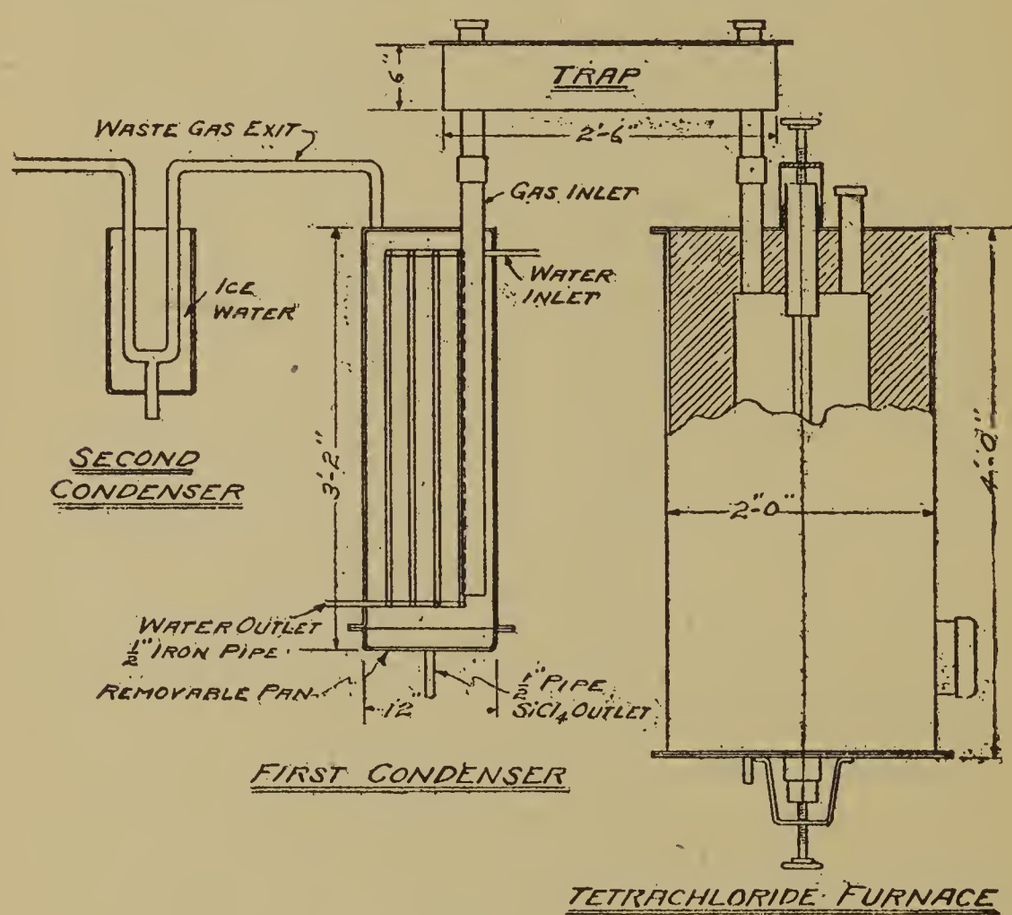


Fig. 2.

temperature of the water entering this condenser was 5° C. The exit temperature was 7° C. The second condenser was maintained at a temperature of —10° C. Ninety-six percent of the silicon tetrachloride recovered condensed in the first condenser and 4 percent in the second condenser.

There was at all times some uncondensed gas issuing from the system. This gas contained some $SiCl_4$ and Cl_2, with possibly some HCl.

It was necessary to charge the furnace every three or four hours with fresh silicon carbide and to discharge the carbon resi-

due. Generally, after charging, power was applied to the furnace for a short time. The carbon resistor lasted for several days before it was necessary to renew it because of oxidation. It was necessary to remove the iron and aluminum chlorides from the trap every 24 hours and the condenser had to be cleaned every three or four days. The chlorine efficiency varied considerably at different times due to operating conditions; the maximum efficiency obtained was about 94 percent. It should be possible to maintain an average efficiency of 85-90 percent. Analyses of the carbon residue indicated that 80 percent or more of the silicon could be extracted from the silicon carbide. A special run made for the purpose indicated that about 93 percent of the $SiCl_4$ made could be condensed in the condenser used.

Measurements were made during different runs to determine the heat evolved by the condensation of the $SiCl_4$ gas and cooling the liquid to about 20° C., or the temperature of the condenser. These figures varied widely, due, it is believed, to difference in temperature of the gases entering the condensers.

The following determinations were made:

175 B. T. U. per lb. $SiCl_4$ (97 cal. per gram)
136 B. T. U. per lb. $SiCl_4$ (76 cal. per gram)

About 1,500 lb. (680 kg.) of silicon tetrachloride was made in the above furnace, and while the data obtained lack the certainty of that obtained by large-scale operation, it is believed that the information presented is sufficiently accurate to be used in the design of a large plant.

No particular difficulties arose in handling silicon tetrachloride, the effect of small doses upon the human system being very similar to that of chlorine. The liquid has a considerable vapor pressure at ordinary temperatures, which makes handling the material exposed to the air disagreeable.

Silicon tetrachloride was used by the Government in the Navy smoke-funnel, in portable smoke apparatus and in hand grenades. It was shipped in pressure cylinders ten inches (25 cm.) in diameter by six feet (1.8 m.) long, both ends of the cylinder being concave. A valve of the same design as used for chlorine was fitted into the cylinder. It was necessary for the seat of the valve to be made of pure lead, as a mixture of antimony and lead was

quickly attacked by silicon tetrachloride. The process of filling the cylinders was to remove the valve and allow 180 pounds (82 kg.) of silicon tetrachloride to flow into the container. The valve was then screwed into place and 21¼ pounds (9.7 kg.) of carbon dioxide forced into the cylinder. The pressure within the cylinder when fully loaded was 300 pounds per square inch (20 atmospheres).

As regards the cost of producing silicon tetrachloride, there are so many variable factors that a close estimate cannot be made. However, it is believed that under normal conditions, with liquid chlorine at 8 cents per lb. (17.6 cents per kg.), the actual cost of manufacture would lie somewhere between 15 and 18 cents per lb. (33-40 cents per kg.) for the crude product.

Comparatively little has been done to find commercial applications for silicon tetrachloride, and the subject is worthy of full investigation.

It is claimed (Troost & Hautefeiulle: Compt. rend. (1871), **73**, 563) that the compound Si_2Cl_6 can be formed by passing $SiCl_4$ over metallic silicon, heated to 1,000° C., or over, according to the equation $Si + 3SiCl_4 \leftrightarrows 2Si_2Cl_6$. It is further claimed that below 1,000° C., but above 350° C., the compound Si_2Cl_6 is unstable and breaks down into silicon and silicon tetrachloride.

We investigated this reaction to study the possibilities of coating metallic articles with silicon, by passing silicon tetrachloride over metallic silicon heated to about 1,400° C., and allowing the gas to pass over iron wires maintained at 300° C. to 800° C. Brown amorphous silicon was deposited about the wires, but in no case was a satisfactory coating obtained.

A quantity of redistilled silicon tetrachloride was submitted to Prof. F. J. Moore, of the Massachusetts Institute of Technology. Under his direction Miss R. M. Thomas investigated the possibilities of using this compound as a condensing agent in a few organic syntheses. The results as they appear below were not encouraging but appear sufficiently interesting to warrant including in this paper:

"Silicon tetrachloride cannot be substituted for aluminum chloride in the Friedel-Crafts synthesis. The following condensations which go readily with the latter failed completely:

$$3C_6H_6 + CCl_4(SiCl_4) = (C_6H_5)_3CCl + 3HCl.$$
$$C_6H_6 + C_6H_5COCl(SiCl_4) = C_6H_5.CO.C_6H_5 + HCl.$$

"These reactions were tried both at room and at higher temperatures and with and without carbon bisulphide as a solvent.

"Silicon tetrachloride does not seem to replace sulphuric acid as a condensing agent. For instance, both m- and p- nitrobenzaldehyde condense with benzene in the presence of sulphuric acid according to the reaction—

$$C_6H_4(NO_2)CHO + 2C_6H_6 = C_6H_4(NO_2).CH(C_6H_5)_2 + H_2O.$$

This did not occur when silicon tetrachloride was used.

"Silicon tetrachloride does behave more or less like zinc chloride or phosphorus oxychloride. In the reaction between dimethylaniline and benzaldehyde the leuco-base of malachite green was obtained. This reaction went without a solvent or in 95 percent alcohol. The latter method (4 g. benzaldehyde, 10 g. dimethylaniline, 4 g. alcohol, 7.5 g. silicon tetrachloride) after heating five hours in a water bath at 70° C., with a return condenser, gave about 70 percent yield of the base. A zinc chloride fusion gives about a 90 percent yield.

"Dimethylaniline and benzal chloride in presence of silicon tetrachloride gave the leuco-base of malachite green but the reaction was not nearly complete after three hours heating at 85° C.

"The reaction between dimethylaniline and Michler's ketone, with phosphorus oxychloride as a condensing agent, gave a quantitative yield of crystal violet when heated on a boiling water bath for five hours. When silicon tetrachloride was used as a condensing agent the yield was greatly diminished.

"After 13 hours at 80-90°, of the materials used (5 g. Michler's ketone, 12.5 g. dimethylaniline, 5 g. $SiCl_4$) one-half remained unchanged.

"In a sealed tube at 100° for 5 hours, followed by one hour at 125°, the yield was increased, but about a third was unchanged.

"With a larger proportion of the chloride (1 g. Michler's ketone, 2.5 g. dimethylaniline, 3 g. $SiCl_4$) at 80° for 12 hours, the yield was about the same as in the second experiment.

"Dimethylaniline did not react with benzophenone in the presence of the chloride.

"Dimethylaniline reacted with phthalic anhydride, in presence of the chloride, but here, also, the action was very incomplete.

"After 7 hours at 80-85°, considerable phthalic acid was recovered and only a trace of product, purple with alkali, was obtained.

"After heating in a sealed tube for 7 hours at 100°, nearly all the phthalic acid was recovered, and not enough other product could be isolated for identification.

"Silicon tetrachloride reacts readily with alcohols, phenols and amines, a fact which would likely limit its use as a condensing agent."

These results indicate that silicon tetrachloride is in no way superior to the better-known reagents for these organic purposes and could be used only if available at a much lower price. It has been suggested that silicon tetrachloride, owing to the ease with which it can be transported, might in some cases be used advantageously as a source of hydrochloric acid.

"After the war" uses of silicon tetrachloride have not yet developed so far as is known. There appears to be a very promising field for investigation along the lines of introducing silicon into organic compounds, and it might well repay chlorine manufacturers who have a prospective surplus of chlorine to seriously investigate this field and develop a use for this easily manufactured chemical which contains over 83 percent chlorine.

Research Laboratory,
The Carborundum Company,
February, 1919.

DISCUSSION.

J. W. Richards[1]: I would like to ask Mr. Hutchins about the determination given on page 317 regarding the heat which is given out in the condensation of the silicon tetrachloride.

The determinations are as follows:

"175 B.T.U. per lb. $SiCl_4$ (97 cal. per gram.)
136 B.T.U. per lb. $SiCl_4$ (76 cal. per gram.)"

[1] Prof. of Metallurgy, Lehigh University.

Was that under varying conditions of temperature of silicon tetrachloride coming into the condenser and varying temperatures of the condenser, or were they precise conditions?

O. HUTCHINS: We got those figures by measuring the temperature of the water used in the condensation. We know those figures exactly and they were the same in every case within one or two degrees, but the temperature of the incoming gases probably varied.

J. W. RICHARDS: Did you not take those?

O. HUTCHINS: We could not get those very well, except that we knew that the maximum temperature was around 250° C.

J. W. RICHARDS: There is a discrepancy between the figure and the maximum heat given by O'Shea, which is 37 calories per gram.

O. HUTCHINS: That is at boiling temperature?

J. W. RICHARDS: Yes, at the normal boiling point.

O. HUTCHINS: On page 316, I say, "The maximum temperature of the gases entering the first condenser was 200° C."

J. W. RICHARDS: If we knew the specific heat of the vapor we could make allowance for its sensible heat.

O. HUTCHINS: Knowing that figure, we calculated the volume and it coincides closely with these figures.

A. C. NEISH[2]: Does silicon tetrachloride and aqua ammonia give a greater volume than, say, anhydrous hydrogen chloride and anhydrous ammonia?

O. HUTCHINS: Yes, it does. There are several compounds formed in this reaction that have obscuring power such as $SiO_2.NH_4Cl$, and $SiCl_4.6NH_3$. This will probably be discussed in the paper by G. A. Richter.

Regarding Mr. Richard's discussion concerning the condensation of $SiCl_4$, would say that we calculated the heat given out in condensing one pound of the gas at 218° C. and cooling the liquid to 20° C. This calculated figure was 125 B.T.U., which agrees fairly well with one of the determinations given in the paper.

[2] Columbia University, New York City.

Presented as part of a Symposium on "Released Electrochemical Information" at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

THE ROLE PLAYED BY SILICON AND TITANIUM TETRACHLORIDES DURING THE PAST WAR.[1]

By G. A. Richter.[2]

Abstract.

A brief statement of the physical properties of these chlorides, followed by a detailed study of their reaction with moisture, or with ammonia gas, or with both, to produce smoke clouds. The order of mixing is of importance to obtain the maximum screening effect. Description also of ship apparatus and trench apparatus used for producing smoke clouds with these chlorides.—[J. W. R.]

Titanium tetrachloride and silicon tetrachloride have played relatively important parts in the programmes of the Navy Ordnance and the Army Ordnance during the past two years. Neither of these substances had any great industrial significance prior to recent hostilities, and whether their use in commercial quantities has stopped with the signing of the armistice depends largely on future efforts in the research laboratories.

The main use to which these chemicals are put in modern warfare is in the production of smoke clouds for concealing or spotting purposes. It is the object of this paper to review the chemical and physical characteristics which make them peculiarly adaptable in this field and to describe the development of the working apparatus which allowed their most efficient use in smoke screens.

Properties of Silicon Tetrachloride.

Silicon tetrachloride when pure is a colorless liquid fuming in the air as a result of reaction with atmospheric moisture. It has

[1] Published by permission of the Director of Chemical Warfare Service.

[2] Res. Chem. Engineer, Brown Co., Berlin, N. H.

a boiling point of 59° C. and a critical temperature of about 230° C. At ordinary temperatures it does not react seriously with oxygen, carbon, zinc, copper or nickel. In the presence of water hydrolysis takes place rapidly. In the absence of moisture silicon tetrachloride reacts with ammonia, forming an addition product $SiCl_4.6NH_3$.

Properties of Titanium Tetrachloride.

No attempt will be made to list a complete summary of the physical and chemical characteristics of titanium tetrachloride. Only those properties which directly concern work discussed in this paper will be given. It is a colorless liquid, fuming strongly in moist atmosphere, and boils at about 138° C. Titanium tetrachloride exposed to moist air goes over into an addition product represented by $TiCl_4.5H_2O$. Excess moisture causes this product to hydrolyze to hydrochloric acid and titanium hydroxide. With dry ammonia there is formed a similar compound to that obtained with silicon tetrachloride.

GENERAL SMOKE FORMING PROPERTIES.

Advantage is taken of the fact that both the silicon and the titanium tetrachlorides hydrolyze readily, forming hydrochloric acid and a residuum of the corresponding oxide or hydroxide. The speed of such a hydrolysis can be controlled. It is a well-known fact that a quantity of the titanium chloride left exposed to the action of moist air will fume for several hours, the time depending on the surface exposed and the humidity of the atmosphere. If the same amount of the tetrachloride is suddenly dispersed by means of an explosive charge, there results a much larger cloud for a correspondingly shorter period. The same is true of silicon tetrachloride, although the cloud produced is of less intensity. In the presence of ammonia the clouds obtained with both the tetrachlorides are intensified even more, due to the formation of ammonium chloride.

When smokes were first demanded by the fighting forces, the Ordnance Departments furnished quantities of phosphorus. Very few actual laboratory measurements or field trials were made during this period of emergency, and having decided somewhat arbitrarily that the white and red varieties of phosphorus were

supreme as cloud producers, a cry was made for greater and greater output. It was soon realized that other more available smoke mixtures must be developed in order to satisfy the demands made by the army and navy. The natural query put by the military man concerning possible substitutes necessitated a quantitative method for determining the value of a new cloud producer and comparing the smoke produced with the well-known heavy screen obtained from phosphorus.

After a long series of experiments, Woodward and Wilson, of the Chemical Warfare Service, established a scale whereby the relative values of different smokes may be expressed quantitatively. The unit adopted is called the Total Obscuring Power, or in abbreviated form T. O. P. The T. O. P. of a given cloud is defined as the product of the volume of smoke in cubic feet produced from one pound weight of substance, by the density of the smoke. The density is the reciprocal of the depth of smoke layer, in feet, beyond which it is impossible clearly to distinguish the filament of a 40-watt Mazda lamp. In other words, the landscape which can be obscured by a pound of the smoke T. O. P. represents the theoretical number of square feet of landscape which can be obscured by a pound of the smoke material.

The first apparatus used in making these determinations consisted of an observation box constructed of wood provided with a movable rod running through it to which is attached the Mazda lamp. The smoke to be tested is generated and passed through the box. The density of the smoke is determined by moving the lamp by means of the rod until the obscuration point is reached. Knowing the density of the cloud and the volume of the chamber, the T. O. P. is calculated. Later in the work when it became necessary to handle greater volumes of smoke, a larger box which contained 228 cubic feet (6.4 cub. meters) was constructed. This larger chamber was provided with a circulating fan which gave a uniform texture of smoke.

When tested in the apparatus described above, phosphorus gives a maximum T. O. P. ranging from 3,000 to 6,000, depending on the humidity of the atmosphere. Since phosphorus was the first substance used in smoke warfare, these values obtained as a result

of burning it in the test box were used somewhat as a standard, the T. O. P. of other smokes produced in other ways being compared with the figures obtained with phosphorus.

SILICON TETRACHLORIDE AS A SMOKE PRODUCER.

Silicon tetrachloride when exposed as such to the atmosphere has very little obscuring power except under very humid conditions. When dispersed in the air by means of an explosive charge, it hydrolyzes quickly, forming hydrochloric acid, but the residual particles of silicon hydroxide, although extremely small, give almost no obscuration. The hydrochloric acid particles formed do produce a screen effect, but it is generally short-lived because of evaporation. On very moist days it is more persistent. No extensive use has been made of silicon tetrachloride as a smoke producer by any of the contending armies except in connection with other chemicals.

Early work had indicated that hydrochloric acid gas when reacting with ammonia vapors could under certain conditions give a smoke having a T. O. P. ranging as high as 1900. The difficulty of handling large quantities of hydrochloric acid as such made it necessary to employ other means. The speed with which the chloride of silicon dissociates to hydrochloric acid and the ease with which it may be handled made it assume an important rôle in further work along this line. Moreover, its low boiling point allows a rapid evaporation when atomized into fine droplets. Such evaporation increases the surface exposed to the moisture of the atmosphere and thus accelerates the formation of hydrochloric acid to a still greater degree.

A series of experiments were conducted in the T. O. P. apparatus varying the proportions of silicon tetrachloride and ammonia; this was supplemented by a series varying the humidity of the atmosphere without disturbing other factors involved in the production of smoke. Later, when more was known concerning the behavior of the reacting substances and the resulting cloud, tests were carried out to determine the effect of a prior dilution of the reacting vapors with air. Wilson and Woodward found that such pre-dilution increased the obscuring power of the cloud formed

in a marked degree. A co-ordination of their results showed that in order to obtain the greatest T. O. P. the following conditions are necessary:

(1) The liquid ammonia and the silicon tetrachloride must be sprayed into the atmosphere in a finely atomized state.

(2) The smoke apparatus must allow a great dilution of both ingredients with air before they are allowed to mix.

(3) There should be sufficient moisture in the air to allow a complete hydrolysis of the silicon tetrachloride before the vapors come in contact with the ammonia vapors.

By carrying out the first requirement, there is exposed to the atmosphere a large surface of liquid, and this results in rapid evaporation. This is important in preventing the liquid from falling to the ground, where a subsequent hydrolysis would not add to the effectiveness of the cloud produced.

As stated before, a previous dilution of each of the vapors with air gives a greater T. O. P. This is not only true in the test chamber but is very evident on the field. Such an increased T. O. P. may be explained by the formation of much smaller particles of ammonium chloride. It is reasonable to expect that with a greater subdivision of mass there should result a greater obscuring effect. It is, moreover, conceivable that beyond a limiting dilution of vapors the total obscuring power will again diminish, owing to an incomplete reaction of the highly diluted gases. Such a maximum dilution was actually obtained in the laboratory experiments, but is much more difficult to realize in the devices used for smoke production in the field.

It is desirable that all of the silicon tetrachloride hydrolyze before coming in contact with the ammonia vapors. Otherwise there is a formation of $SiCl_4.6NH_3$, which in itself is not readily hydrolyzed. Even under the best operating conditions it is found necessary to reduce the ratio of silicon tetrachloride to ammonia from the theoretical value of 2.5:1 to 2:1, in order to avoid an acid smoke. Acidity may be explained either by the formation of the complex $SiCl_4.6NH_3$ which may on subsequent hydrolysis cause an acid atmosphere, or by the absorption of ammonia by particles of silicon dioxide formed in the main reaction. Operating conditions which allow the formation of the complex mentioned tend to lower the T. O. P. obtained.

It is also possible to obtain a reduction in the T. O. P. of a smoke by forming it in the presence of an excess of moisture, as rain or steam. This is probably due to the formation of larger droplets of hydrochloric acid solution and consequently larger particles of ammonium chloride.

The development of the Navy Smoke Funnel was a more or less direct application of the principles and facts established in accordance with the above paragraphs. The funnel is a smoke device attached to the deck of a vessel and operated to produce a smoke screen as a protection against submarine attacks. It consists of a horizontal cylinder about six feet (1.8 m.) long and twenty-eight inches (70 cm.) in diameter, provided with suitable connections for the introduction of silicon tetrachloride and ammonia under most advantageous conditions. One end of the cylinder is equipped with a hand fan, the function of which is to furnish large volumes of air for dilution of the respective vapors before mixing. The silicon tetrachloride is vaporized through a bank of very efficient spray nozzles located about 12 inches (30 cm.) in front of the fan and directed toward the other end of the cylinder. The ammonia is introduced through another spray nozzle located about 12 inches (30 cm.) from the exit end of the cylinder and also pointed toward this end of the apparatus.

The ammonia is forced through the nozzle from a steel cylinder by means of its own vapor pressure. The silicon tetrachloride is propelled from a second steel cylinder by means of dissolved carbon dioxide. The tetrachloride line running from the cylinder to the nozzles is provided with a glass-wool strainer to remove any particles of sediment which may accumulate in the system on standing. To operate this smoke device, both valves leading from the tetrachloride and the ammonia reservoir are opened wide and the hand fan at the rear of the funnel revolved in order to provide sufficient air dilution of the vapors.

Dry carbon dioxide was selected as a propellant for the silicon tetrachloride for several reasons. It does not react with the tetrachloride at any temperature likely to be encountered. The silicon tetrachloride is forced through the nozzles at nearly a constant pressure and rate, whereas with compressed air or nitrogen the rate of discharge of liquid is seriously affected by the progressive drop in the pressure within the container. Such a change in rate

of discharge of silicon tetrachloride causes a variation in the proportion of chemicals delivered and consequently lowers the obscuring power of the smoke produced.

When the pressure of a solution of carbon dioxide in silicon tetrachloride is released, it effervesces in a way similar to soda water. This behavior apparently causes an exploding effect on the particles of solution discharged through the nozzles, thus resulting in added disintegration and more rapid vaporization.

Inasmuch as the vapor pressure of liquid ammonia and the total pressure of the silicon tetrachloride solution rise and fall with temperature changes, they tend to keep pace in rates of discharge through the nozzles. Experimentation established an optimum mixture of twelve percent of carbon dioxide in the silicon tetrachloride. Under a surveillance temperature of 60° C. this composition will generate a pressure of 600 pounds to the square inch (40 atmospheres). At —26° C. this pressure drops to 225 pounds (15 at.). These experiments were conducted in a vessel two-thirds full of the liquid mixture.

Every effort is made to duplicate as closely as possible in a practical way those conditions which proved most efficient in the laboratory. A ratio of two parts of tetrachloride to one of ammonia is maintained without much trouble. The liquids are sprayed through the funnel at a rate of seven pounds (3.2 kg.) and three and one-half pounds (1.6 kg.) per minute respectively. A heavy white cloud of high obscuring power is formed. It is easily set up, readily controlled and is so nearly neutral in composition as to allow the vessel generating the smoke to pass through it. A single pair of cylinders contains sufficient material to generate an efficient cloud for a period of thirty minutes.

The smoke knapsack devised for trench warfare employs the same principles incorporated in the specifications of the navy smoke funnel. The apparatus consists of a pair of thin sheet steel tanks about six inches (15 cm.) in diameter and twenty-six inches (65 cm.) long. One tank serves as a container for the ammonia and the other for the silicon tetrachloride. They are joined in the form of a knapsack and provided with connections to spray the liquid contents of the cylinders through a set of nozzles. The nozzles are placed some distance apart, but no effort is made to realize air dilution by means of a fan or other

artificial device. The flow of chemicals is readily controlled by the operator. Enough material can be put in one outfit weighing sixty pounds (27 kg.) to generate a low-hanging cloud of high obscuring power for a period of about fifteen minutes. With proper regulation of flow the smoke produced is neither harmful or obnoxious. It was calculated that two such portable smoke knapsacks would be sufficient to screen the movements of an entire company in open formation.

As stated in an earlier part of this paper, silicon tetrachloride filled in a steel shell and exploded by means of a T. N. T. booster gives very little smoke on a dry day. The cloud improves greatly in moist weather, but even under optimum conditions it does not compare favorably with sulphur trioxide or phosphorus.

TITANIUM TETRACHLORIDE AS A SMOKE PRODUCER.

It has been demonstrated experimentally that when exposed to perfectly dry air, titanium tetrachloride does not form a cloud or mist. In the presence of small amounts of moisture there is formed the addition product $TiCl_4.5H_2O$, which has a high obscuring power. An excess of moisture causes this complex to hydrolyze, forming hydrochloric acid and the hydroxide of titanium which has a low obscuring power.

Just as in the case with silicon tetrachloride it was found that a much better cloud could be formed by the titanium tetrachloride reacting with ammonia under suitable conditions. To obtain the highest total obscuring power in this way, it is necessary to cause a previous dilution of the two vapors with air and to allow the titanium tetrachloride sufficient time to hydrolyze as completely as possible before mixing with the ammonia. Otherwise, there is produced a complex of the tetrachloride and the ammonia which is not easily broken down and which lowers the T. O. P. of the cloud.

Laboratory investigation yielded a maximum T. O. P. with the ammonia-titanium tetrachloride mixture about fifty percent higher than that produced with the silicon compound. It is, however, more difficult to handle through spray nozzles, and is less volatile than the silicon tetrachloride compound. Trial runs made on navy vessels indicated that the overall efficiency obtained with the

titanium tetrachloride was no greater than that realized with the silicon compound, and that the slowness of evaporation of the titanium tetrachloride caused gumming of the nozzles and a general nuisance on board ship. It was never used in the smoke funnel manufactured for the American navy.

For artillery shell where it is impracticable to have two compartments for two liquids neither the titanium nor the silicon tetrachloride is of much value. In an average atmosphere the titanium smoke is superior to the silicon cloud but in very humid air the silicon cloud outranks the smoke produced from an equal volume of titanium tetrachloride. This may be explained by the formation of the fairly efficient $TiCl_4.6H_2O$, which in the presence of more water hydrolyzes to HCl, thus losing much of its obscuring power.

The titanium tetrachloride has been used to a limited extent in hand grenades, but for screening purposes does not compare with phosphorus or the zinc chloride combustion mixture which was developed in this country. In mopping up after a frontal attack, dugout grenades filled with tin tetrachloride were used in large numbers. Various attempts were made to substitute the titanium chloride for the less available tin compound but these never proved successful.

In summing up the relative values of silicon tetrachloride and titanium tetrachloride as smoke producers, it is evident that the silicon compound is by far more important so far as American developments are concerned. Neither tetrachloride was used extensively except in combination with other materials, especially liquid ammonia. The low cost of manufacture added advantages to the silicon tetrachloride. Had the war continued it is very probable that both tetrachlorides would have been used in newer devices which were under development.

DISCUSSION.

F. A. LIDBURY:[1] I should like to ask whether there is any information available as to the optimum proportions of these two constituents of smoke, the ammonia and chloride?

[1] Works Mgr., Oldbury Electrochemical Co., Niagara Falls, N Y.

W. D. BANCROFT:[2] That has been worked out but it is not given in this particular paper and I have not the exact figures. The only thing I remember offhand is that you do not use equivalent quantities of ammonia and hydrochloric acid. There is apparently an adsorption of hydrochloric acid or ammonia, I don't remember which, by the silica set free, and consequently your smoke has not the acidity you would expect it to have if you simply wrote formulas for it.

F. A. LIDBURY: The reason I asked that question is that my impression was that the formation of a double chloride, of a compound of ammonium chloride and silicon or titanium chloride, as the case may be, had a good deal to do with opacity values.

W. D. BANCROFT: I am quite sure that is not right at all; we have a whole set of data on the thing and it is simply a question of a few percents extra of one or the other, but it does seem certain in the case of titanium tetrachloride that when you are using that by itself, the best opacity is obtained when you form a hydrate of titanium tetrachloride with water and afterwards when you hydrolyze that, it is not so good. That does not come up in the case of the silicon tetrachloride, and I am quite sure that there is no satisfactory evidence to show that the smoke is due to the formation of any double compound whatever in there.

R. E. WILSON:[3] I can answer the question by stating that there was no double compound formed under the conditions prevailing in our smoke generators. The smoke was made by first passing moist air through silicon tetrachloride vapor, which hydrolyzed the silicon tetrachloride into hydrochloric acid and silicon hydroxide (silicic acid). The hydrochloric acid thus produced reacts with the ammonia to give ammonium chloride. The optimum obscuring power per unit weight of materials was obtained by using about five molecules of ammonia to one of silicon tetrachloride, which corresponds to five of ammonia to four HCl. The exact reason for this is not definitely understood, though it may be that some of the ammonia reacts with the silicic acid or forms some additional compound.

[2] Prof. of Physical Chemistry, Cornell University.

[3] Asst. Director, Res. Lab. of Applied Chemistry, Mass. Inst. of Technology.

MR. DODD: I would like to ask how sulphur chloride ammonia works? It works very well, apparently.

R. E. WILSON: They are much less satisfactory; under any considerable dilution they do not tend to react; that is in high dilution such as is necessary to get finely divided particles. The dilution in the case of most smokes is from 50 to 400 volumes of air per unit volume of gas. This gives very fine, stable particles of high obscuring power, and sulphur chloride will not react under these conditions.

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THE PREPARATION OF FLUORINE.[1]

By W. L. Argo, F. C. Mathers,[2] B. Humiston,[3] and C. O. Anderson.

Abstract.

1. Fluorine has been prepared in quantity by the electrolysis of fused potassium hydrogen fluoride with a copper cathode and a graphite anode.

2. Fluorine was prepared by the electrolysis of anhydrous hydrofluoric acid rendered conducting with potassium hydrogen fluoride. Both the Moissan cell and the cell as modified by Meslans were tried. This method is altogether impracticable for preparing fluorine in quantity, since the platinum anode is attacked rapidly by that element.

3. Only traces of fluorine were obtained by heating lead tetrafluoride. A method was found for the ready preparation of lead tetrafluoride. Fusing lead peroxide with potassium hydrogen fluoride was found to be a very satisfactory method for the preparation of lead tetrafluoride.

4. Anhydrous hydrofluoric acid was prepared by heating sodium hydrogen fluoride. This salt was found preferable in many ways to the potassium salt formerly used.

THE PREPARATION OF FLUORINE BY HEATING LEAD TETRAFLUORIDE.

Brauner[4] prepared fluorine as early as 1882 by heating the compounds $CeF_4.H_2O$ and $3KF.2CeF_4.2H_2O$. Later[5] he found that potassium fluoplumbate, $3KF.HF.PbF_4$, gave fluorine on heating. Hydrofluoric acid only was evolved below 230°, but on

[1] Published by permission of the Director of the Chemical Warfare Service.
[2] Professor of Chemistry, University of Indiana.
[3] Chemist, Jackson Laboratories, Deepwater Point, N. J.
[4] Jour. Chem. Soc. (1882), **41**, 68.
[5] Jour. Chem. Soc. (1894), **65**, 392; Zeit. anorg. Chem. (1894), **7**, 1.

heating the salt in a platinum tube below a red heat, a gas was obtained which possessed the odor of fluorine, liberated iodine from iodides and attacked silicon with great rapidity. Ruff[6] repeated Brauner's experiments but was unable to confirm his conclusions, possibly because of failure to reproduce Brauner's experimental conditions. Smythe, Rosenthal, Ray and Banta at the American University Experiment Station have prepared the colorless fluorides of bromine and iodine by heating a salt of lead fluoride. Accordingly experiments were undertaken to find an efficient method of preparing lead tetrafluoride, since it seemed that this might be the solution of the problem of obtaining fluorine in quantity.

Brauner[7] obtained the salt of lead tetrafluoride, $3KF.HF.PbF_4$, in the following ways:

1. The freshly precipitated oxide $Pb_5O_7.3H_2O$ was treated with a mixture of KHF_2 and hydrofluoric acid.

2. Potassium hydroxide and lead peroxide were fused together in the proportion of three of the former and one of the latter; and the mass obtained was moistened with water and added slowly to a large excess of pure strong hydrofluoric acid.

3. One mol of lead tetra-acetate was treated with three mols of KHF_2 and an excess of hydrofluoric acid. Crystals of the double lead tetrafluoride appeared on evaporation.

Brauner prepared PbF_4 from the double salt by grinding the latter to a fine powder and throwing it onto the surface of cold concentrated sulphuric acid. A pale yellow solution first formed, which set to a jelly from which PbF_4 deposited on heating. This experiment could not be confirmed by Ruff.

The method most generally employed for the preparation of the double salt is to treat potassium or sodium plumbate with aqueous hydrofluoric acid. In this way both the potassium and sodium double salts were prepared by Smythe, Rosenthal, Ray and Banta.

Elbs and Nubling[8] found that plumbic salts were produced during the electrolysis of certain concentrated acids using a lead anode, providing the current density was maintained high and the

[6] Zeit. anorg. Chem. (1907), **12**, 17; Zeit. anorg. Chem. (1916), **98**, 27.
[7] Jour. Chem. Soc. (1882), **41**, 68.
[8] Zeit. Elektrochemie (1903), **9**, 776

temperature low. They had carried out no experiments with hydrofluoric acid, and it was thought that this method might serve for the preparation of lead tetrafluoride. Accordingly a cell was constructed consisting of a wax electrolyzing vessel surrounded by a cooling mixture of ice and salt; with electrodes of lead, and an electrolyte of varying concentrations of hydrofluoric acid. With 50 percent hydrofluoric acid and a current density of 1.5 amperes per square decimeter, no gas was liberated at the anode. At 3 amperes per square decimeter a little white precipitate (presumably lead difluoride) formed on the anode, but no black lead peroxide. A similar result was observed with all current densities up to 14 amperes per square decimeter. No lead peroxide was formed nor could a test for tetravalent lead be obtained in the solution. The above experiments were repeated with 96 percent hydrofluoric acid, with essentially the same results as obtained with the more dilute acid.

Lead peroxide and red lead are not soluble in and apparently are not acted on by 50 percent hydrofluoric acid. However, the 96 percent acid acts readily both on red lead and on freshly precipitated lead peroxide, with the formation of some lead tetrafluoride in each case. Red lead is attacked more readily by hydrofluoric acid than is the peroxide; but it is objectionable because of the large amount of acid wasted in forming lead difluoride. That some tetrafluoride was formed was shown by the hydrolysis into lead peroxide on the addition of water.

Dry lead peroxide (Eimer and Amend C. P.) was apparently unattacked by 96 percent hydrofluoric acid, even on standing several days. Freshly prepared lead peroxide dissolved without difficulty in the concentrated acid. Red lead was treated with an excess of nitric acid and the lead peroxide was washed with water and then dried. The undried paste was not soluble when four times the theoretical weight of hydrofluoric acid was used. This was probably due to the dilution by the water contained in the paste. The importance of having the acid highly concentrated is shown by the rapid hydrolysis produced by the addition of very little water to the solution of lead peroxide. Some water may be added if potassium acid fluoride or ammonium fluoride is first added. This is due to the formation of a double salt with the lead tetrafluoride in solution. A yellowish white residue remained

after the action of the hydrofluoric acid was complete. In the experiments, 50 grams of PbO_2 were treated with 35 grams of hydrofluoric acid and allowed to stand at least 12 hours. In one case the insoluble white residue weighed 19 grams, hence, counting the material that dissolved as tetrafluoride, the yield was approximately 62 percent. The most noticeable thing was the great tendency of lead tetrafluoride to hydrolyze even in 25 percent acid, if the solution was boiled. The low yields of lead tetrafluoride obtained by earlier experimenters were in all probability due to the use of too dilute acid. Had they treated the specially prepared and dried sodium or potassium plumbate with almost anhydrous acid instead of the stock 50 percent solution, the yields might have been much greater. However, experiments along this line were discontinued because a much easier method of preparing the salt was found.

Lead peroxide reacts with fused acid potassium fluoride with the formation of lead tetrafluoride. The reaction was carried out in either of two ways: the materials were mixed prior to heating or the lead peroxide was added slowly to the already fused salt, at a temperature ranging from 250° to 300°. The reaction was carried out in a copper vessel. The heating was continued with constant stirring until the black color disappeared and the mass became gray. In these experiments 219 grams KHF_2 (containing 2 percent of water) and 164 grams of lead peroxide (containing 93.5 percent PbO_2) were fused. These were theoretical quantities on the basis that all the HF goes to form PbF_4. The course of the reaction was followed by the gas (H_2O) evolution. The mass finally became semi-solid, since the final product melts higher than potassium acid fluoride. The difference in weight between the sum of the weights of PbO_2 and the fused (anhydrous) KHF_2 and that of the final product was 22 grams, whereas 25 grams is the theoretical loss of water in the reaction. The analysis of the final product showed 37.5 percent of PbF_4 which is 73 percent of the theory. To prepare pure crystals of the material it was ground finely, leached in warm hydrofluoric acid solution, filtered and allowed to crystallize.

The product obtained by the above process can be used directly in those experiments requiring the liberation of fluorine by heat. Only relatively small amounts of the element were obtained by

this method. For this reason experiments along these lines were discontinued. The method above described is a general one that may serve for the preparation of other perfluorides. Manganese tetrafluoride was readily prepared in this way.

THE PREPARATION OF FLUORINE BY ELECTROLYSIS.

Fluorine was first prepared in quantity by H. Moissan[9] by the electrolysis of anhydrous hydrofluoric acid rendered conducting

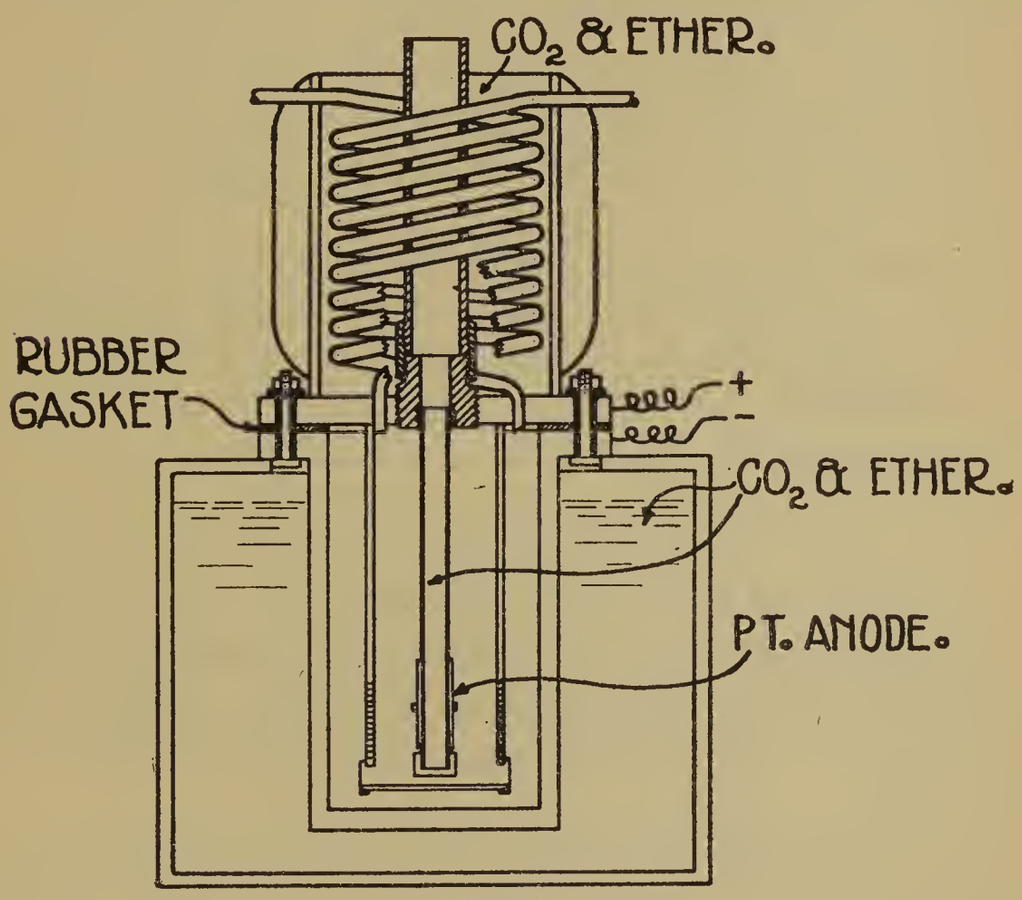

FIG. 1.

by the addition of potassium hydrogen fluoride. Moissan began with a platinum U-tube as his electrolyzing vessel, the openings in which were closed with fluorite stoppers in which the electrodes of platinum-iridium were set. Later he found that copper would serve equally well as an electrolyzing vessel, but that platinum electrodes must be used, as in Fig. 1. With copper electrodes, polarization took place due to the formation of a non-conducting film of copper fluoride on the anode. Owing to the heat generated in the electrolysis and the high vapor pressure of hydrofluoric

[9] Comptes rendus. (1886), **102**, 1543; (1886), **103**, 202, 256, 850; (1889), **109**, 362, 637.

acid under usual laboratory conditions, it was necessary to keep the cell well cooled. In his earlier experiments, Moissan effected the cooling by the evaporation of chlormethane. Later he used acetone containing solid carbon dioxide. Other investigators have used calcium chloride freezing mixture[10] and alcohol and solid carbon dioxide[11].

(a) Preparation of Anhydrous Hydrofluoric Acid: The usual method of preparing anhydrous hydrofluoric acid was to heat potassium hydrogen fluoride.[12] Moissan decomposed the anhydrous salt in a platinum retort. Ruff and Plato,[13] and Goldschmidt[14] replaced the platinum retort by a copper one and found that this latter was not acted on appreciably. Ray and Banta confirmed this result.

Two explosions occurred at this laboratory while distilling the hydrofluoric acid from the double salt. The cause of the explosions was in both cases a clogging up of the copper tubing leading to the condenser with particles of sodium fluoride carried along by the gaseous hydrofluoric acid. To avoid this, a copper retort was fitted with a distilling column an inch (2.5 cm.) in diameter which was closed with a brass cap that served as a safety valve. Two tubes entered the distilling column near the top; one leading to the copper condenser, the other serving to admit air through a sulphuric acid wash bottle. The condenser was joined by copper connections to a receiver also protected by a wash bottle of sulphuric acid.

A very slow stream of air was caused to pass through the apparatus during the distillation. The absence of bubbles indicated that the pipe line was obstructed. The receiver and the condenser were cooled by an ice-salt mixture.

Experiments were made with the Moissan and with the Meslans type of cell, using platinum anodes. While some fluorine was obtained, the platinum anodes were attacked even though the temperature of the cell was kept at —30 to —60°. It was quite evident that the corrosion of the platinum anode was so great that fluorine could not be made commercially in this way. An Acheson

[10] Ruff and Ipsen: Ber. deutsch. chem. Ges. (1903), **36**, 1777.
[11] Cuthbertson and Prideaux: Phil. Trans. Roy. Soc. (1905), **205**, 319.
[12] Fremy: Ann. Chim. Phys. (1836), [3] **47**, 6.
[13] Ber. deutsch. chem. Ges. (1904), **37**, 673.
[14] Monatshefte (1897), **28**, 297.

graphite anode containing 98 percent graphite was tried; but it swelled and disintegrated even when no current was passing. The action appeared to be due to the hydrofluoric acid rather than to fluorine.

Since potassium hydrogen fluoride melts at approximately 200°, and decomposes but slightly at this temperature, it was determined to try to obtain fluorine by electrolyzing the fused salt. It was found that a graphite anode could be used. The final form of

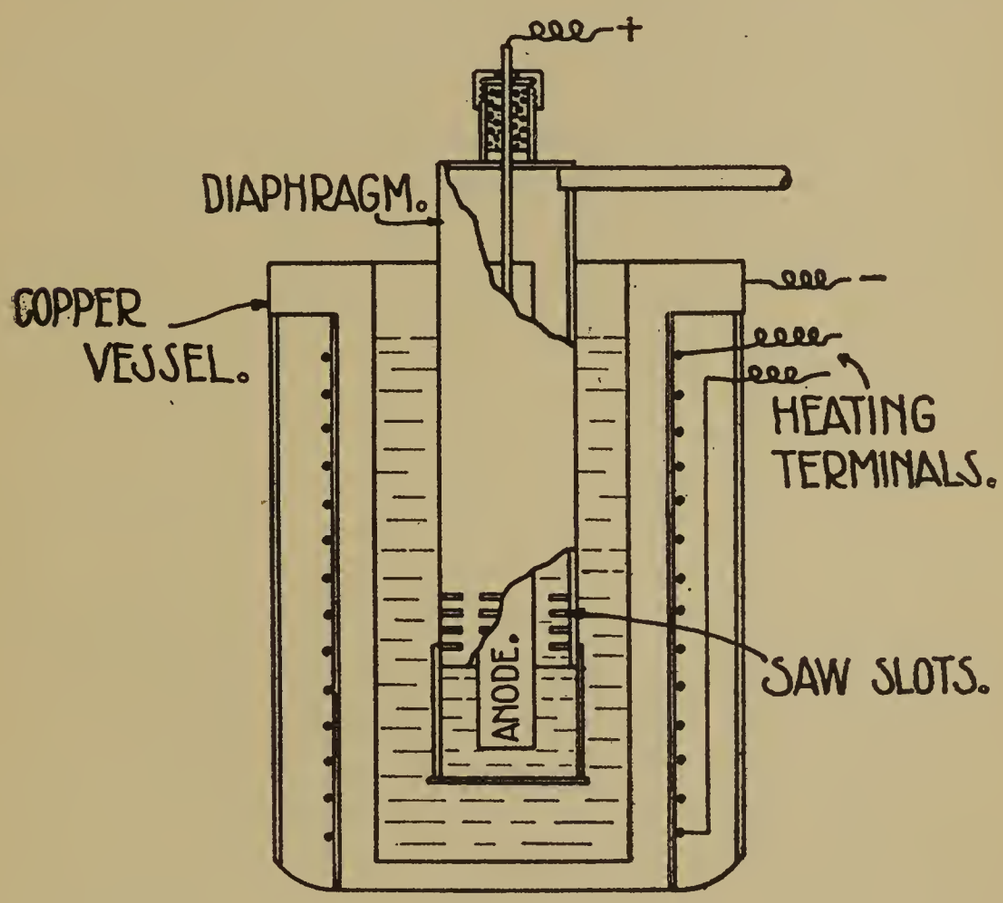

FIG. 2.

the cell is shown in Fig. 2. The electrolyzing vessel which served as cathode was of heavy copper, 3½ inches (9 cm.) internal diameter and 8 inches (20 cm.) deep, surmounted by a heavy flange. The cell was wound with ni-chrome wire insulated from the walls with asbestos paper. To prevent loss of heat by radiation the wires were surrounded by asbestos wool and asbestos paper. The diaphragm was two inches (5 cm.) in diameter. Near the bottom a number of slots were cut for the purpose of decreasing the voltage of the cell. Hydrogen evolved from the bottom of the electrolyzing vessel during electrolysis was prevented from rising to

the inside of the diaphragm where it would unite with fluorine, by means of a false bottom which deflected the bubbles. Considerable space remained below the anode for potassium fluoride to collect. The graphite anode was insulated by packing it in a mass of finely powdered fluorspar, held closely and firmly round a copper bolt by means of a packing gland and nut. The copper lead was held in place by a washer of fluorspar while the finely powdered mineral was pounded in. Finally, a fiber washer was placed on top and the packing nut screwed down securely. The graphite anode was attached to the copper terminal by boring a hole in the former and pounding it firmly into the latter.

The fused bath has a slight solvent action on copper vessels, so that there is a gradual accumulation of copper fluoride in the bath. The solvent action increases with rising temperature, and when there is a localized overheating such as is obtained with a Bunsen burner, it takes but a short time for a hole to be eaten through the vessel at the point it is heated. The local disintegration of the vessels has been overcome in large measure by electrical heating. In an electrically heated cell containing approximately 1,500 grams of bath about 20 grams of CuF was formed in a week.

A fresh bath, prepared from potassium fluoride and commercial hydrofluoric acid, always contained some water which had to be electrolyzed out at a low current density before fluorine was obtained. If the current density was too high, polarization resulted and the voltage rose to 50 or 60 volts. There was sparking through the oxygen gas film, suffusing the bath with a blue luminescence. A current density of 2 to 3 amperes per square decimeter was passed until a test of the anode gas showed the presence of fluorine, after which a much higher current was used without polarization resulting.

A difficulty which proved very troublesome indeed for a time was the foaming of the bath, particularly around the anode. This frothing carried up the electrolyte into the exit tube of the diaphragm where it froze and clogged up the system. Frothing resulted only after the bath had been electrolyzed for a considerable length of time so that the electrolyte had become viscous from the accumulation of potassium fluoride and copper fluoride. To prevent it, all that was necessary was to add hydrofluoric acid and

regenerate the bath. Before it was found that regeneration would solve the difficulty, stirring the bath and shaking the anode by a motor were tried without avail. Simply grinding up the salt and adding hydrofluoric acid without filtering the solution to remove impurities serves to put the bath in good shape so far as frothing goes.

Since the electrolytic decomposition of potassium hydrogen fluoride results in a loss of hydrofluoric acid and the consequent formation of potassium fluoride it is necessary to regenerate the bath from time to time. Inasmuch as the potassium fluoride is not molten at the temperature of the bath, it gradually settles to the bottom of the electrolyzing vessel. Attempts have been made to run anhydrous hydrofluoric acid into the bath under pressure, but these have not proven successful since the acid is a gas and it merely bubbles through the molten double salt and reacts but slowly, if at all, with the solid on the bottom of the cell. Furthermore, in view of the fact that the bath dissolves copper slightly it is necessary occasionally to dissolve the electrolyte in water and filter off the impurities. During this operation the hydrofluoric acid can be added to the cell.

The procedure followed in this laboratory has been as follows: After using the bath until it becomes quite viscous as a result of the presence of considerable solid KF it is poured into a copper dish or on a sheet of copper. After cooling, the salt is crushed fine, placed in a copper vessel, and commercial hydrofluoric acid added. As a rule, the salt is considered as all KF and enough acid is added to form KHF_2. Inasmuch as there is always considerable KHF_2 present in the bath when poured out, a reasonable excess of acid is ensured. The bath with its excess of acid is heated slowly for several hours and finally brought to 225°, when it is easily molten. It is then returned to the cell and a current of 2 to 3 amperes per square decimeter is passed for several hours to electrolyze out the water. It has been noticed that as the last traces of water are removed a temperature of from 240° to 250° is necessary to keep the electrolyte molten.

In order to prevent contamination of the bath by copper and the consequent necessity for dissolving up and filtering off the bath, a Dixon graphite crucible was substituted for the copper electrolyzing vessel. It was rapidly attacked, however, owing to

the high percentage of clay in its make-up. A vessel made from graphite boards was next tried but the structure was such that it absorbed a large amount of hydrofluoric acid and so proved unsatisfactory. It is proposed to try a crucible of electrode graphite such as is used for the anode as soon as one can be secured. If this proves satisfactory the regeneration may be effected in the cell, without much difficulty. If anhydrous acid is used instead of the commercial product the period of drying and preliminary electrolysis to remove water will be materially shortened.

In the first cells used the voltage was 18 to 20 volts for a current of 3 amperes. This seemed higher than necessary. Accordingly the vessel was used as the cathode, thereby decreasing the current density of this electrode; and the anode was dropped an inch (2.5 cm.) below the diaphragm. With this arrangement it was found possible to pass 10 amperes at 12 volts. At a temperature of 240° to 250° a voltage of 15 was usual with a current of 10 amperes. At higher temperature the conductivity of the bath increased appreciably, but loss of hydrofluoric acid offset this advantage.

Working with ten amperes at 12 to 15 volts, and a temperature of 225°-250°, the current efficiency is about 70 percent.

Among the tests for fluorine may be mentioned the following:

1. The not unpleasant odor of small quantities of fluorine is very characteristic. The odor reminds one somewhat of a mixture of ozone and chlorine but without the irritating action of the latter. This test readily serves for detecting leaks in the apparatus. Apparently fluorine possesses but little toxic action, although long exposure to rather large quantities produces headache.

2. The unlighted gas mixture from a Bunsen burner is instantly ignited by fluorine. This test is delicate, but less so than the odor.

3. Sulphur is ignited quickly by the gas. A soft charcoal, *e. g.*, a piece of charred wood, glows brilliantly. Selenium reacts with fusion. Chloroform vapors react readily.

On account of the difficulties involved in the preparation of thoroughly dry acid potassium fluoride it occurred to us that the anhydrous acid could be prepared with considerably less difficulty

if acid sodium fluoride were substituted for the potassium salt usually employed. The following are among the more noteworthy advantages:

1. Sodium fluoride is not hygroscopic and hence overdrying of the double salt does not leave a material that has to be protected from the air; potassium fluoride, on the other hand, is very hygroscopic indeed.

2. The sodium salt decomposes below the fusion temperature, while the potassium salt does not.

3. The sodium salt is cheaper.

4. The sodium salt contains a higher percentage of available hydrofluoric acid.

Two grams of sodium fluoride were treated with an excess of aqueous hydrofluoric acid. The reaction took place rapidly with evolution of heat. The excess hydrofluoric acid was driven off, the product was dried at 260° for a few minutes and weighed. The gain in weight corresponded to 23.6 percent hydrofluoric acid. On heating again for a few minutes to 260°, a further loss of hydrofluoric acid to 20.9 percent was found. It was evident that 260° was too high a temperature for drying, so the experiments were repeated at a temperature of 140° to 160°. The percentage of hydrofluoric acid taken up was 32.3, which is theory. This percentage was reduced to 32.14 by drying a second time, at 140° to 160°.

This work was done in the catalytic laboratory of the Research Division of the Chemical Warfare Service, under the general direction of W. D. Bancroft.

DISCUSSION.

F. A. J. FitzGerald:[1] One of the most interesting things Col. Bancroft brought out, is this attack of graphite by hydrofluoric acid. Why does it occur? We have got the fact, but I would like to hear the theory of it.

[1] The FitzGerald Laboratories, Inc., Niagara Falls, N. Y.

W. D. BANCROFT:[2] I suppose it would be perfectly simple to work out a theory for it, but we were busy doing other things and we did not do anything with it. I may add that there is one possibility from an analytical point of view which Professor Hulett is carrying out now, and that is to see whether he can use fluorine as a means of discriminating between carbon and graphite. I imagine he will have the same difficulties as with all other tests, that as he passes from carbon to graphite, there will be a range within which he will have to use his judgment or prejudices to decide whether it is graphite or carbon. On the other hand, it will give another method of differentiation, and he can say that when fluorine attacks the material under certain conditions he will call it carbon, and when it attacks it under other conditions, he will call it graphite.

C. A. DOREMUS:[3] There are one or two things in Moissan's attempt to which I wish to call attention. He did not begin his work with this reaction but chose other compounds and finally set up an apparatus for electrolysis. He used cork for a stopper and noticed that the cork was carbonized. In his monograph on fluorine, he states that this fact was the first hopeful indication in three years of research. He then went on and made the apparatus which Professor Bancroft has described for the electrolysis of potassium acid fluoride, at a low temperature. The isolation of the fluorine was a matter of such importance that it was told to the French Academy and a commission was appointed to witness the experiment. Berthelot was one of the members of that commission. Moissan at that time had the use only of a lecture room, a lecture table at odd hours where he could make demonstrations, and knowing the importance of this experiment, he prepared everything with extreme care and set up the apparatus, and when the commission arrived, what he predicted was to take place did not take place, there was no electrolysis and no evolution of fluorine; the anhydrous hydrofluoric acid he found later was not a conductor, and the reason Moissan had previously obtained his fluorine was that there was a little potassium fluoride which sprayed over into the acid. He made a fresh preparation with a little potassium fluoride later and there was no difficulty in his

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[3] Consulting Chemist, New York City.

securing fluorine. Professor Bancroft has mentioned the experiments which may be shown in public. The most interesting one was one Moissan made with Professor Dewar in London, where fluorine was allowed to impinge on liquid hydrogen and the two inflamed spontaneously.

F. C. FRARY:[4] I would like to ask whether the electrolysis of a fused mixture of sodium and potassium acid fluorides was tried? This would fuse at a lower point than either constituent, and therefore would be more available for the experiment.

W. D. BANCROFT: That was not tried. The experience with the fluorine was just like the experience with everything else in the war service—the armistice was signed too soon.

F. C. MATHERS[5] (*Communicated*): In order to credit the portions of this work to the proper individuals, this paper or record should show that the broad fundamental patent on the process of preparing fluorine by the electrolysis of a fused bath, which employees of the United States may secure under certain restrictions, is to be given to Mathers. A second patent for apparatus is to be given jointly to Mathers and Humiston.

The disintegration of Acheson graphite anodes in anhydrous hydrofluoric acid containing potassium hydrogen fluoride (the electrolyte of Moissan) is very rapid. Electrolysis for a few minutes causes the graphite to flake or scale off in pieces much as if water soaked porous material had been allowed to freeze. Ordinary foundry coke or hard electrode carbon was more resistant.

Ruff found rhodium to be the most resistant of all the platinum metals to hot fluorine gas in a tube. Therefore it seems that rhodium would have made the best electrode, but the purchasing department was unable to get material for trying this experiment. Of course there is a big difference in conditions between the action of hot fluorine in a tube and nascent fluorine in anhydrous hydrofluoric acid at low temperatures. This difference is shown by the fact that graphite was a satisfactory anode material in the fused bath but not in the anhydrous hydrofluoric acid. A possible explanation is that the graphite, which is porous, is not wet by the fused bath while the liquid hydrofluoric acid fills all of its capil-

[4] Director of Research, Aluminum Co. of America, New Kensington, Pa.
[5] Associate Prof. of Chemistry, University of Indiana.

laries. It was very unfortunate that rhodium could not have been tried. Moissan had found that iridium or platinum-iridium alloy was less satisfactory than pure platinum. In some of his later experiments he used club shaped platinum anodes, for the reason that they lasted longer. However no particular mention was made by Moissan or by Ruff that the anodes were severely attacked. This can be accounted for by the fact that they prepared only small quantities of fluorine—the making of a few grams of fluorine probably would not produce a very noticeable attack on the anodes. Concerning the Brauner method of making fluorine by the heating of lead tetrafluoride, Ruff[6] obtained sufficient fluorine to detect it by odor but not to ignite gas, hence he concluded that it was merely an interesting method of no value.

Ruff and his students at the University of Berlin have made many experiments on fluorides and have always prepared the free fluorine by the Moissan method. It is strange that in the years covered by these experiments, he did not find this much easier method of making fluorine. Ruff made the statement that the preparation of fluorine would always remain a difficult laboratory operation due largely to the trouble of making and of handling anhydrous hydrofluoric acid. Here again the use of sodium acid fluoride instead of the potassium acid fluoride would have made this operation easier. If copper apparatus is used in the Moissan method the hydrofluoric acid must be very concentrated. If not almost anhydrous the copper in contact with the anode dissolves and is reprecipitated as a sponge upon the cathode whereby a short-circuit is soon produced. The protecting or insulating film is formed on the copper only when the concentration of the hydrofluoric acid is high. This solubility of the copper in the aqueous hydrofluoric acid prevents the dehydration of the electrolyte, if much water is present, by the simple scheme of electrolytic decomposition of the water whereby ozone is evolved at the anode. No fluorine is produced until all of the water is gone. The current efficiency of the fused bath was determined by bubbling the fluorine through aqueous sodium hydroxide and measuring the oxygen thus produced in glass tubes. Ordinary foundry coke or hard electrode carbon can be used as anodes in the fused bath. Silver, gold, platinum, nickel, copper, stellite and nichrome, as anodes,

[6] Z. Angew. Chem. (1917), II, 1275.

were all rapidly attacked. The copper was much the most resistant. Copper in contact with graphite is slowly attacked and fluorine is evolved. In some of the early experiments fluorine was made by using in the fused bath an inverted copper funnel inside of which a piece of graphite was fastened by copper wire. The anode wire was connected directly to the funnel. All of the other metals and alloys given above were rapidly attacked even when in contact with graphite and no fluorine was given off. Two grams of sheet copper fastened to a graphite anode lost 0.07 gram in an hour. 3.8 gm. of gold lost 0.36 gm. in a few minutes.

A paper presented at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

CHEMICAL WAR SECRETS AND RELEASING MANUFACTURERS' RIGHTS[1]

By Edward Gudeman.[2]

Abstract.

A discussion of what attitude the Government and the Government chemist should now take towards valuable chemical information divulged to the Government by individuals and corporations for the national benefit during the progress of the war.

We are all aware that most chemical manufacturers, individuals and corporations, during the time of the war, not alone placed their services at the disposal of our Government, but many turned over their plants, making no attempt to retain for themselves any information they possessed in their specific lines. In many cases salaries were guaranteed to those entering Government employ, making good differences between Government remuneration and former civic salaries.

In connection with such action, those in control of chemical manufacturing industries gave the Government their own exclusive private information, their methods and trade secrets. It is true that in many cases these secrets could have been called aloud from the housetops without divulging anything new or novel, but in many cases they were of value and meant commercial advantages to the concerns possessing them. Some of these secrets were patentable and in other cases were tricks of the trade. What they were and their specific value is immaterial, they were property rights, often exclusively owned and well guarded by the concerns that divulged them. Some may even have been imagi-

[1] Manuscript received January 30, 1919. Read also at the Chicago Meeting of the American Institute of Chemical Engineers, December, 1918.

[2] Consulting Chemical Engineer, Chicago, Ill.

nary, mental idiosyncrasies or Chemical Christian Science methods, such as the peculiar location of a piece of machinery, applying power from the right or the left, using a specific indicator, or taking a sample just as the clock strikes one. The fact remains that all the information possessed by some concerns was freely and fully given to Government departments, such as the Bureau of Mines, Treasury Department, to the Navy officials, and mainly to the Army—the Chemical War Service Branch. These bureaus confiscated many of our chemists, and they took not only the cream of chemists, but the whole milk, leaving only the "has-beens" to get under cover in other war activities.

What is the Government going to do with this mass of valuable information, not alone that which was freely given—more properly loaned— and that which the Government obtained during the last two years while operating chemical industries, often with the help of those captains of chemistry who temporarily left civic chemical industry for Khaki and Blue? What as to the individual chemist, who while directly or indirectly working for the Government obtained perhaps much more chemical information than he gave, and who today has much more chemical information than he gave up, directly due to his having had the privilege of wearing the uniform with the chemical insignia?

We cannot expect him to forget all of this, nor have we any way to make him forget the knowledge he so obtained. Is he justified in making use of this information and in divulging it? I believe he is, if he uses it *ethically*. He has a perfect right to offer his services to those who consider the information he has of value, but always offering his services first to his former employer, if he had one, giving him the first opportunity to re-employ him. I hold that he has no ethical or moral right to try to exploit this information by trying to induce anyone outside of existing chemical industry to enter it, using this information as the foundation for building up competing chemical industry. We may not be able to prevent such unethical practice, but need anticipate very little of it, as the chemists capable of exploiting knowledge obtained in that way are of such value to existing industry that they need not go outside thereof to reap rewards or to make their marks.

But what will the Government, as such, do with the fund of

information obtained from existing chemical industries and with the knowledge it obtained while conducting chemical business? Any information or knowledge pertaining to chemical industries *actually non-existing* in our country before the war can be made public, as no American chemical industry will be effected.

We have the other two phases to consider, the first being where a "running" chemical concern was, perhaps *in toto,* taken over by the Government, operated by the Government or under its control or supervision. In such a case the Government obtained full information how the plant was and should be operated, from the previous owners, who in many cases became Government employees. The second phase is where the Government was a party to operating a running plant and where with the Government's assistance new processes or methods were worked out or new products were originated. In this case the Government has an interest in the plant, there exists here a Government right in the products, process, and new knowledge obtained. There is no necessity to draw too fine a line between these two phases, the one considering the exclusive property rights of the pre-war owners and the other considering the rights of the Government, due to its quasi-partnership. Under the term property rights we must consider the concrete process and product and also the abstract knowledge and information. What should be done to protect chemical industries, chemical manufacturers and the public as to its share in the Government rights? How and to what extent should this "secret" and industrial information be released to, or preserved for, or withheld from, the public at large?

It has been intimated that when the Chemical War Service Branch goes out of existence, all industrial information, scientific and practical, given the Government, and all the information of similar kind obtained by the Government in its own research and experimental work, is to be boxed up and stored away in the vaults of the Army Department, ready for any future call. This, no doubt, would fully protect the chemical manufacturers, as far as their secrets and tricks of the trade are concerned, but it would at the same time bury a lot of very valuable information, which will be useless when the emergency calls it up again. Would it not be feasible for the Army Department to create a new bureau to replace the Chemical War Service Branch, such bureau to have

the right to make any information available to those chemical manufacturers specifically interested therein? The giving of this information could be restricted to those manufacturers who assisted the Government, directly or otherwise, in obtaining any such information. This would put all loyal American chemical manufacturers into the same boat, would be of great value to chemical industry, would create no hardship for anyone, and would not deprive the public of any benefits. No secrets or tricks of the trade would be divulged, and information obtained from an exclusive source would be withheld from others. Information divulged to the Government would be protected, and would not prevent pre-war exclusive owners from obtaining patent rights; in fact could be used to prove patent validity and priority. This would require a change in our patent acts, and might lead to patent acts giving proper protection to a patentee, as we all know that the most valuable patents today have lawsuits attached to them, often making them a liability instead of an asset to the inventor. It has been suggested, and there seems to be strong agitation in favor thereof, to have the Government make public, throw wide open, everything done during the war, without consideration as to property rights of any chemical manufacturer. If the suggestion to publish a complete history of what chemistry has done for the United States, without consideration of its effect on chemical industry is carried through, so as to make such volumes of interest, they should be printed in Greek, Chinese or Aramaic letters, as this would convey to the public, men and women, as much valueless information as if this history were printed in English letters. We can rest safe, as no doubt the best men and women still in the Chemical War Service Branch will return to pre-war chemical industries, and those left, if any, to write this official chemical war history, can do little harm to chemical industries or chemical manufacturers. Also such a history would have to be distributed free, under congressional frank perhaps, as few would be willing to even waste postage thereon. It would be waste of paper and ink, but some could be utilized for shaving, curl or other purposes. It is also certain that by the time this so-called chemical history appears, it will be ready to file away as ancient, obsolete, antiquated chemical lore.

We must do something to protect our chemical industries and

the manufacturers, and prevent that which they consider as valuable, given, really loaned, to our Government from being spread broadcast, becoming public property. We must stand back of them, and enable them to have a strong sayso as to how this industrial information, their property rights, are to be protected, even if we have to force into the background some chemical individuals who are trying to plume themselves with peacock feathers and thereby shine in reflected glory. Our Government can solve this problem from the ethical and moral viewpoint, and no doubt will do so, but it can do no harm for us to place into its line of vision a few chemical sign posts, marked—manufacturers' rights —private information—no trespass.

Presented as part of a Symposium on "Released Electrochemical Information" at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

PORTABLE ELECTRICAL FILTER FOR SMOKES AND BACTERIA.[1]

By Arthur B. Lamb[2], G. L. Wendt[3] and Robert E. Wilson.[4]

Abstract.

Description of a (portable Cottrell) precipitator, intended to be carried by one man, to protect him against smoke. The apparatus devised was efficient, but expensive, and was never put into actual service. It was found to kill bacteria in the air passing through it, which suggests its possible use in plague hospitals. It might also be useful in some industrial conditions.—[J. W. R.]

INTRODUCTION.

The adequate protection of the respiratory system of the soldier in gas warfare involved two distinct methods of air purification. The first consisted in the removal of toxic gases from the inhaled air by chemical absorption or physical adsorption during its passage through a canister filled with granular absorbing materials. This was popularly supposed to be the only type of protection afforded by the ordinary mask. As a matter of fact, however, such granular absorbing materials are unable to remove suspended particles of toxic solids or liquids which are too large to diffuse rapidly, as do gases, and yet too small to settle out or be removed by centrifugal action in passing through the layers of absorbent granules. A second method of purification was therefore required for the removal of such particles, and accordingly every mask was equipped with some form of filtering material. This filtering material was invariably placed before the absorbent because the solid or liquid particles caught on the filter frequently vaporize slowly, giving off toxic gases which must be absorbed by the chemicals in the canister before reaching the wearer.

[1] Published by permission of the Director, Chemical Warfare Service.
[2] Harvard College, Cambridge, Mass.
[3] Asst. Prof. of Chemistry, Univ. of Chicago.
[4] Mass. Inst. of Technology, Cambridge, Mass.

Smokes generated from stannic chloride, silicon tetrachloride, titanium tetrachloride, oleum and burning phosphorus have been used very extensively since the early stages of the war, both as protective screens and because of their irritating effect on the nose and throat of the enemy. The latter use was relatively unimportant, as, in the first place, the irritation produced was comparatively slight, and in the second place, thin layers of cotton padding placed in the box respirator gave complete protection against them without introducing any serious obstacle in breathing.

When, however, the Germans introduced the so-called "sneeze gas," consisting usually of diphenylchlorarsine, a much more effective protection was required. Not only is this substance extremely toxic, but a smoke produced from it by detonation in a shell consists of unusually fine particles and is extremely penetrating, so that an ordinary filter is quite useless against it.

Faced by this danger, great efforts were made to secure effective and satisfactory filters. It at once became evident that there was no serious difficulty in securing effective filters, filters which would remove completely even as penetrating a smoke as that of diphenylchlorarsine. But while such filters afforded complete protection, most of them were extremely fine pored and impermeable, and imposed a high resistance to breathing, so that their use in a portable equipment for men in active movement was quite out of the question. Nevertheless, as a result of extensive and intensive study, practicable filtering materials were finally devised which gave satisfactory protection against these smokes and only increased the resistance of the gas mask about one inch of water at the excessive breathing rate of 85 liters per minute.

Even this added resistance, however, is undesirable. Indeed, it represents a 75-100 percent increase in the resistance of the most recent type of gas mask. Moreover, the possibility of still more penetrating smokes still remains, which would require filters of correspondingly higher resistance for complete protection. Under these conditions an energetic investigation of the possibility of utilizing the Cottrell method of electrical precipitation in portable individual units was undertaken, for this device seemed to offer the only hope for a complete removal of smoke, coupled or combined with a negligible resistance to breathing.

The difficulties in the way of such a development were by no

means underestimated. The Cottrell installations employ an extremely high potential, indeed, one of 100,000 volts or thereabouts. Could this potential be reduced to such a point so that a small portable power plant would be feasible? Could such an equipment be made sufficiently robust to function continuously, not deteriorate on standing, be easy and cheap to make, and withal not weigh over three or four pounds? This outlook was certainly far from promising, but as a matter of fact, a fairly satisfactory solution was secured.

This problem, like so many others of the period, was a contest with gravitation. In other words, it was a problem of weight reduction; the construction of the lightest possible precipitator and the lightest possible power plant to operate it. No attempt will be made to record at all fully the many possibilities which were tested, nor the many steps which were taken before the final results were achieved. It is desired, however, to acknowledge the great assistance we have received from many outside laboratories, particularly, that from the Research Corporation through Dr. Cottrell and Mr. Meston, from the Westinghouse Electrical and Manufacturing Company through Mr. Skinner, from the General Electric Company through Dr. Whitney, from the Samson Electrical Company through Mr. C. S. Colby and from Mr. Jacobson, of Pittsfield, Mass.

DEVELOPMENT OF APPARATUS.

In the Cottrell precipitator air is passed through large tubes or flues containing a centrally placed fine wire, charged to about 60,000 volts. At this high potential the ions, always present in air, are given so high a velocity that their collisions with other gas molecules result in ejection of electrons from the latter, *i. e.*, result in "ionization by collision." As a consequence the air becomes highly conductive and a "corona" discharge is formed. The electrons attach themselves to the smoke particles, these under the high potential gradient are hurled against the charged walls of the tubes and adhere to them. At first it was hoped that a relatively low potential produced possibly from the galvanopile might be utilized, at least if an independent source of ionization, such as radioactive material could be provided. Considerable progress was made in the construction of a satisfactory galvanopile of thin

copper and zinc strips, but it developed that there was not a sufficient number of ions already present in the air, nor would even a large amount of radio-active material supply an adequate number of these ions to give anything like a complete removal of the smoke with any reasonable potential gradient. Using, for instance, a potential gradient of 2,000 volts per centimeter and an amount of radium-C equivalent to 120 milligrams of radium, less than 10 percent of a standard smoke at a rate of passage of 10 liters per minute was precipitated. These experiments demonstrated that the "corona" discharge was the only method of ionization which offered any hope of furnishing the great number of ions undoubtedly required for the rapid removal of the relatively heavy smoke particles in the very short intervals of time available in a precipitator of practicable size.

Initial experiments using an induction coil as a source of high potential resulted in failure. Careful study showed that the failure was due to two causes. In the first place, the coil gave an alternating discharge and the alternations were so rapid that the charged particles, which are relatively heavy and immobile, did not undergo appreciable movement or precipitation. In the second place, the form of the discharge wave from the induction coil shows an exceedingly high and narrow peak voltage, the ratio of maximum to effective voltage being between four and ten with various types of coil. This means that a precipitator which will not allow a spark from the peak voltage will also not allow a corona discharge from the average effective voltage.

The first difficulty was obviated by making use of the rectifying action of a point against a surface. If the inner or high potential electrode is made to bristle with small sharp points and is centrally suspended within a metallic tube of not less than two centimeters diameter this rectification is practically complete and the current may be made to pass from the inner electrode only, so that practically all deposition of the smoke is on the inner walls of the tube.

The second difficulty was avoided by interposing a layer of dielectric material between the electrodes, of such a thickness that its dielectric strength is equal to the difference between the maximum and the effective voltage of the induction coils to be used. Since this is usually five or six thousand volts, only a thin layer

of dielectric is required to prevent sparking completely. The dielectric does not appreciably interfere with the corona discharge. Its use has the further advantage that it removes the necessity of absolute central alignment of the inner electrode; if the latter is slightly off center or if some of the points project farther from its surface than others no harm follows, whereas without the dielectric the discharge is concentrated at these points, with great loss in precipitating power.

When these two features were incorporated in the precipitator it was found that the induction coil is a nearly ideal source of potential and all further work was in this direction. The precipitator in its present form is thus very simple.

As many tubes as are necessary for the air volume required are used, connected in parallel. In its present form the precipitator uses four glass tubes 1½ centimeters in diameter and about 12 centimeters long, each wrapped in a strip of copper gauze. In a form for quality production these glass tubes would presumably be replaced by thin tubes of iron or aluminum, glazed or enameled on their inner surface. The inner electrodes are made by cutting a strip of copper wire cloth of fine mesh in such a way that only two adjacent parallel strands are included, together with the correspondingly short lengths of the many crosswise strands cut. This is then twisted lengthwise through four or five axial turns, and the ends of the long wires are inserted into the heavier electrode holders and insulated from the outer electrode tube.

The weight of the precipitator proper is less than a pound, and when manufactured in quantity need not exceed a few ounces.

Numerous tests of commercial induction coils were made, including most of the automobile and marine engine ignition coils. It was found that by far the most efficient coil, for this purpose, at present on the market, is the "Perfex," "all-in-one" model. Other coils delivered a larger absolute amount of energy but at the cost of greater weight or of higher energy consumption in the primary circuit. The "Perfex" coil operates continuously for at least ten hours, as shown by actual test, on a primary current of ½ ampere at 6 volts.

Mr. E. B. Jacobson, of Pittsfield, Mass., submitted an experimental coil in which the core iron was coiled back and ended in the base of the vibrator. In this manner the magnetic circuit was

nearly closed, the vibrator serving as the last line in the circuit. This coil proved somewhat superior to the "Perfex" coil in its efficiency, *i. e.,* in ratio of energy delivered in the secondary to that received in the primary circuit. A modified "Perfex" coil was then constructed in which a similar principle was adopted, namely, that the iron wires of the core were made twice as long as in the first type and were coiled back around the outside of the secondary winding and ended in the base of the vibrator. The results obtained with this coil were on the whole about as good as those obtained with the Jacobson coil. The actual behavior of these three coils can be seen in the following table:

Comparison of Induction Coils.

Induction Coil	Input		Output		Efficiency	
	Volts	Amp.	Mean Voltage	Microamp.	Watts	Percent
"Perfix" No. 1.	5.8	0.50	650	14	0.0091	0.31
Jacobson	"	0.58	750	16	0.012	0.36
"Perfix" No. 2.	"	0.46	875	16	0.014	0.52

The "Perfex" coil No. 2 is evidently better under these conditions. Further tests seemed to indicate that the Samson operates more effectively at lower current consumption, while at higher amperage in the primary circuit the Jacobson is the more effective. The "Perfex" coil is being manufactured on a large scale and the modification involved in the new form is slight, so that it offers but little difficulty with regard to production. The weight of all these coils is about the same; namely, about 400 grams or 14 ounces. This weight can be somewhat reduced by proper minor modification in the housing and casing.

As a primary source of current, special light-weight dry cells were considered, but the necessity for a substantially constant voltage to operate the induction coil eliminated them from consideration. The lead storage battery in a special form, developed for airplane work, appears to offer the best possibility so far as absolute weight efficiency is concerned. The manufacturers are confident that a suitable battery can be provided furnishing 30 watt-hours, *i. e.,*

the required ½ ampere for 10 hours with a weight of about 3 pounds. A small commercial battery furnishing ½ ampere at six volts for 14 hours has been used in the actual operation of the precipitator with entire satisfaction. It weighs a little less than six pounds. It was not specially designed for light weight. For industrial purposes, where light weight is not so important a prerequisite, the alkali storage battery would undoubtedly be superior because of its more robust nature. Batteries of this sort are already available on the market, furnishing sufficient energy for 10 hours' use and weighing less than six pounds.

EFFICIENCY OF FINAL APPARATUS.

In testing different types of precipitators, coils, etc., it was necessary to use a fairly accurate quantitative method for measuring the smoke precipitation efficiency. A majority of the tests were performed by passing through the precipitator at a known rate, a dilute smoke, made by mixing one part of ammonia in 5,000 of air with one part of hydrochloric acid gas in 5,000 of air. This forms a very thin smoke consisting of extremely fine particles. The efficiency of the precipitator was determined by passing the effluent smoke through an alundum crucible immersed in water, which completely removes all remaining smoke, and determining the amount of ammonium chloride in the water by the use of Nessler's reagent. The testing apparatus is rather complicated and need not be described here. Some quantitative tests were also made against SO_3 smoke, and qualitative tests against some of the toxic smokes which would be met on the battlefield. It was soon found, however, the results obtained on ammonium chloride smoke were sufficiently representative of what could be expected with other smokes, and could be measured and reproduced much more accurately than the other tests.

The rate of flow is, of course, a very important variable as affecting the efficiency of the smoke precipitation, the higher the rate of flow the lower being the efficiency. The following set of experiments shows the typical results obtained when testing a single precipitator tube, of the type previously described, against the ammonium chloride smoke. The previously described battery and coil were also employed in the test.

Effect of Rate of Flow on Precipitating Efficiency of Single Tube.

Rate of Flow Liters per Minute	Efficiency Against NH_4Cl Smoke 1 in 10,000
3	100%
5	100%
10	93%
15	85%
25	79%

These results indicated that for a man exercising only moderately four or perhaps five tubes should afford adequate protection against any toxic smoke. A full-size portable outfit was therefore roughly constructed and tested out in the man-testing chamber by wearing it in a cloud of diphenylchlorarsine smoke set up by detonation. Such a test has been found to penetrate all except the very best filters within a few minutes, but the man wearing the electrical precipitator was able to remain for a full hour without suffering any discomfort whatever. This indicated conclusively that the smoke precipitation was complete under these conditions. There was no appreciable resistance to the flow of air through the precipitator. A canister containing gas-mask absorbent was, of course, used between the precipitator and the wearer in order to remove any toxic gas or vapors which might be present.

It also seemed desirable to determine whether or not air-borne bacteria could be removed by the use of this precipitator. Advantage was accordingly taken of the method devised by Major Tolman for setting up a known concentration of air-borne bacteria and drawing the air through various types of filters. Air containing a high concentration of *bacillus prodigiosis* was drawn through a single precipitator tube at a rate of five liters per minute. The issuing air was bubbled through sterile water. Cultures taken from this water showed that the air passing through the precipitator was absolutely free from bacteria. Further tests indicated that the inside of the precipitator tube was also sterile, although the bacteria must have been precipitated therein. The precipitator therefore not only removed the bacteria from the air, but killed them.

It is possible that the death of the bacteria was caused not by the direct action of the precipitator but by the ozone, a small

amount of which is produced by the electrical discharge. Careful tests were made to determine whether the ozone thus produced caused any discomfort to the wearer, but it was found that a very thin layer of charcoal was sufficient to remove entirely this ozone from the air. It was also found that the production of the ozone had no effect on the efficiency of the absorbents against various toxic gases.

The signing of the armistice and the closing down of experimental work in the Chemical Warfare Service unfortunately prevented the final development of the electrical precipitator apparatus in a compact and neat form. Enough was done, however, to demonstrate the entire practicability of the method from a scientific standpoint. It is, of course, practically certain that the large amount of special and expensive equipment needed would prevent the use of this in actual warfare, except possibly in a very limited way, and as a last resort against a very penetrating toxic smoke. It would seem, however, that such a portable precipitator might well be of considerable value for industrial uses, where workers must be protected from smoke, dust, etc., with a minimum of breathing resistance, or in hospitals where the protection of nurses and possibly patients from air-borne bacteria would be very desirable.

DISCUSSION.

A. F. Meston[1]: The deductions of the authors as to the electric conditions present in the portable precipitator are open to question. But inasmuch as the current supplied by the apparatus is so small that it is difficult to use oscillographs or other means commonly employed for analyzing circuit characteristics, it is hardly possible to say what conditions are present.

Now it is not necessary to use direct current or rectified current to produce precipitation. If the electrodes are very close together and the length of the gas path is long, or the velocity of the gas is very low, a collection of the suspended particles can be obtained with alternating current, the frequency of that current being a

[1] Research Corp., New York City.

factor. When you use an induction coil, you have a very peculiar circuit; you have a current the voltage of which is very high when of one polarity and comparatively low when of the other. This has its drawbacks and its advantages. One drawback is that the peak voltage, as the authors point out, is very apt to cause disruptive spark-over before the average voltage is high enough for good precipitation. The advantage is that you do have, in a sense, a rectified current; at least you have a higher electric stress across the gas with one polarity than you do with the other, and the movement of the particles, while probably oscillatory, tends to be for the most part in one direction. To this, I think, is largely due the value of an induction coil for getting electric precipitation.

We tried one of these small precipitators connected to a great many storage batteries in series, and the result was not satisfactory, at least with all types of electrodes. With direct current there was a tendency for the fume, as it collected, to bridge across the electrodes when the distances were small, and at the end of a short run spark-over occurred. The oscillatory nature of the induction coil seemed to prevent this.

If we assume that rectification can be obtained with a point to plate discharge (such as that present in the small precipitator) because of the difference in spark-over potential when the point is negative and when positive, it would be expected that rectification of the current from an induction coil would be easily secured if the peaked side of the current wave was positive and the flat side was negative. But even if this be true, we are yet confronted with the existence of both positive and negative ions acting as carriers for the fume particles, and, also, with the fact that the polarity of the electric field alternates regardless of the current flowing. What goes on, therefore, would seem to be less obvious than the authors' deductions would indicate.

I think that the current passing through the gas in the precipitator is not completely rectified. The gases are ionized by the corona discharge, the ions become attached to the fume particles and the charged particles oscillate back and forth under the stresses of the alternating electric field; but the forces acting are stronger in one direction than the other and the result is that most, but not all, of the fume collects on the surface of the tube. Some collects on the discharge electrode. If the distance between the electrodes

is not very small, the velocity of the gases must be very slow, indeed, in order to obtain good precipitation with alternating current. We usually associate precipitation, when alternating current is used, with agglomeration of the particles, but I am not sure that we should do so. I do not think agglomeration of the particles while suspended necessarily comes in at all.

W. D. BANCROFT[2]: The principle of the Cottrell precipitator was used in another way. They wanted to get a general method for analyzing smokes, as it was rather hopeless to find a special method for each particular smoke; consequently the Research division developed a small form of the Cottrell precipitator, in which they had a central wire and a very light aluminum foil outside of it as the other electrode, and they used a rectified current of about 15,000 volts. Under these circumstances, the whole of the smoke is precipitated on the aluminum foil electrode which can afterwards be weighed, it thus giving a direct gravimetric determination of the amount of smoke in a given volume of air. That worked very satisfactorily indeed, and it had the other advantage that a microscopic examination showed whether the smoke that you were dealing with under these particular circumstances was a liquid or was a solid. In many cases a smoke is generated by vaporization and condensation, and a substance which, in equilibrium is solid at a given temperature, might perfectly well be present in the form of liquid. At one time there was a good deal of question whether diphenylchlorarsine went through the mask as a drop of liquid, or whether it was present as solid grains, so this modification of the Cottrell precipitator enabled the Research Division to answer the question as to the particular form in which the smoke was and to get a quantitative determination of the smoke with very little difficulty. It was a very helpful form of apparatus.

R. E. WILSON[3]: Another suggestion might be offered. Possibly not along quite the same line, but it is such a convenient little trick that it might be valuable in a great many cases. If just behind that precipitator is put a tube of charcoal, it is possible, by getting the increase in weight of the precipitator and the charcoal tube to determine how much suspended solid matter, and how

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[3] Asst. Director, Res. Lab. of Applied Chemistry, Mass. Inst. of Tech.

much condensable vapors are actually present per unit volume of gas. Furthermore, by simply heating up the charcoal under proper conditions, about 200° C. for most materials, the condensed liquid can usually be completely recovered and very easily analyzed.

H. S. TAYLOR[4]: I just want to ask a question for information purposes. It is this: we are engaged on some work using electrical precipitation of solid particles and I wanted to know if anybody here could tell me whether the lighter or the heavier particles fall first. I wanted to know if one can fractionate the particles by means of a Cottrell precipitator?

R. E. WILSON: If you eliminate the action of gravity, the lighter particles will come first, but in most cases you get a combination of gravity and electrostatic action, which usually precipitates the heavier particles first.

W. D. BANCROFT: It seems to me that this question of gravity could be eliminated very readily, and that you probably could get a fractional separation if you ran the things up instead of down; they will stick to the electrode well enough so that you could get a lot of interesting results. Of course in the work that we were doing in Washington, the problem was to get everything out, and they were not working along the lines of fractional precipitation, but I cannot see any earthly reason why fractional precipitation could not be secured.

B. SPEED[5]: There is one phenomenon I have noticed in the Cottrell process that I have never seen applied or recorded anywhere. At the Riverside Cement Works in California, on the roof of the kiln house, the cement dust had collected in lumps, distinctly crystalline lumps, lumps of about the shape of an ordinary copper sulphate crystal, about a quarter of an inch long and sufficient in number so that, scooping your hands over the dust, you could get half a dozen. They were of different sizes. Whether or not that collection and formation of the particles in those crystalline forms—they were easily crushed, they were not very coherent—was due to the fact that the individual smoke particle as a crystal developed some electrical polarity due to

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[5] Head of Physical Lab., Western Electric Co., New York City.

heating it, without any reference to the Cottrell process, I do not know. I merely want to put that on record as a fact, that the smoke that was coming from those kilns did collect and agglomerate in distinctly crystalline groups.

G. L. WENDT[6]: The question as to the rectification of the current is, to be sure, by no means settled. There are too many factors involved. The form of the potential wave varies widely among different induction coils. The coil giving a high peak voltage was of course out of the question. The "hot spark" type in which the average voltage is high, though the peak is relatively low, was desired. Along with this it was desirable to have a coil in which the negative voltage was as low as possible. This seemed to be the case with the Samson coil. In any case, the direct current across the ionization gap could readily be measured by a microammeter, and the average voltage could be determined by means of an electrostatic volt-meter. The figures for the voltage on the secondary coil given in the table were thus obtained. There is decidedly an excess of voltage in the one direction. Finally, the time taken by the smoke to pass through the precipitator is so brief that any hesitation on the part of the smoke particles would result in their failure to precipitate. There was also some deposit on the inner electrode, but the larger quantity by far appeared on the tube itself, and thus indicated a nearly uni-directional current.

[6] Assistant Professor of Chemistry, University of Chicago.

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AN ELECTROLYTIC PROCESS FOR THE PRODUCTION OF SODIUM PERMANGANATE FROM FERROMANGANESE.[1]

By Robert E. Wilson[2] and W. Grenville Horsch.[3]

Abstract.

A description of laboratory experiments followed by semi-plant scale operation, wherein ferro-manganese anodes were used in a diaphragm cell in sodium carbonate solution, with the production of an 8 to 12 percent solution of sodium permanganate in the anode compartment. The anodes gradually accumulated a skin of oxides of iron, manganese and silicon, which caused local heating by its electrical resistance; this was removed every 24 hours by sand-blasting. The solution was evaporated to 30 percent $NaMnO_4$. Details of design, costs and operating cost are given for a plant producing half a ton of $NaMnO_4$ per day, which it was estimated could be produced under war conditions at about 60 cents per pound.—[J. W. R.]

INTRODUCTION.

The investigation which is outlined in this paper was carried on largely by the Research Division of the Chemical Warfare Service, U. S. Army, with headquarters at the American University Experiment Station, Washington, D. C.

The absorbent used in gas mask canisters consisted of a mixture of charcoal and soda-lime. About October, 1917, it was found necessary to put sodium permanganate into the soda-lime, primarily in order to afford some protection against certain oxidizable gases not absorbed by soda-lime. Potassium permanganate was available but was found to be worthless for the purpose.

[1] Published by permission of the Director, Chemical Warfare Service.

[2] Major, Chemical Warfare Service, Washington, D. C.

[3] First Lieutenant, Chemical Warfare Service, Washington, D. C.

A canvass of the country resulted in finding only one manufacturer who could furnish sodium permanganate, and even this was very impure. A new method of oxidation of manganate to permanganate was therefore worked out by the Research Division. Using this modified method, the large-scale production of fairly good sodium permanganate in 30 percent solution was begun about January 1, 1918.

There were still two objections to this product, namely, its very high cost and the presence of a considerable amount of impurities. The high cost was due partly to the war-time market conditions and their effect on raw materials and labor, and partly to the rather complicated series of operations connected with the fusion method of manufacture. The Government had to pay during the war an average price of about $2.50 a pound. As impurities there were present other salts (chiefly NaCl) to the extent of about 25 percent of the permanganate. Although only 3 percent of sodium permanganate was used in the soda-lime, it was definitely shown that the fraction of 1 percent of impurities thus introduced decreased the absorptive properties of the soda-lime, and also accelerated its deterioration.

In view of these conditions it seemed desirable to make a thorough study of other possible ways of manufacturing sodium permanganate. This study indicated that the electrolytic process offered the best prospects. Accordingly, it was decided to investigate the electrolytic process, with the view of lowering the cost of production and getting a purer product.

DEVELOPMENT OF SUITABLE ANODES.

The first problem was to determine the best kind of anodes. Special alloys of manganese with silicon and other metals were tried. Previous work by Dr. Lovelace had shown that such alloys gave high efficiency. They possessed the disadvantage, however, of very high cost. Commercial ferro-manganese was the only readily available material. Experiments tried with small lumps of ferro-manganese showed that a fair yield could be obtained, and the promise was sufficient so that the experiments on anode composition were postponed and attention now directed toward the casting of ferro-manganese.

As is well known, commercial ferro-manganese containing 80 percent manganese is crystalline and very brittle. Everyone consulted on the matter predicted that castings, if they could be made at all, would be too fragile to use, and probably would fall to pieces on cooling. Indeed, the first attempts to re-melt this material in a crucible and to cast it into slabs about 9 by 12 by 1 inches, using a sand mold, resulted in complete failure. The work was done in a bronze foundry, since the American University Experiment Station was not equipped with the necessary apparatus for work of this sort.

At this point it was decided to take the matter up with Dr. Colin G. Fink, of the Chile Exploration Company, to whom is due the credit for the successful solution of the casting problem. He obtained entirely satisfactory anodes by making the castings in the form of long bars, adding certain deoxidizing agents or scavengers, and by slow cooling in kieselguhr (infusorial earth). Some of the castings were poured in sand, some in graphite molds. The graphite produces smoother castings and is to be preferred where a large number of duplicates are to be cast, since the graphite mold may be used over several times. Less than ten percent of the castings made in this way have to be discarded.

DETERMINATION OF BEST OPERATING CONDITIONS.

Casting troubles having been eliminated, attention was at once turned to the determination of the best operating conditions. The most important points to be investigated were anolyte composition, current density, and temperature, in their relation to current efficiency and voltage. The experiments on anode composition were dropped, because the work that had been done indicated that the improvement would not be large, and because, as already mentioned, the cost of special alloys would be much greater than that of commercial ferro-manganese. As catholyte the only possibility is sodium hydroxide solution, since sodium hydroxide is the product formed at the cathode during electrolysis. From the early experiments it was found that the concentration of the catholyte had a negligible, if any, effect on the current efficiency, but a solution of maximum conductivity (about 18 percent NaOH) should be used in order to reduce the cell voltage as much as possible. In all of these condition experiments small

diaphragm cells were used, with a cooling coil placed in the anode compartment for controlling the temperature. The cathode was a piece of sheet iron.

Composition of Anolyte: Various solutions were tried out as anolyte but only two salts were found which gave at all satisfactory results, namely, sodium hydroxide and sodium carbonate. If pure sodium hydroxide solution of fairly high concentration is employed as the initial anolyte, manganate is formed at first instead of permanganate. The manganate may be oxidized subsequently when the concentration of free base becomes low. There is, however, one very serious objection to using hydroxide solution. At concentrations in the vicinity of two percent sodium hydroxide the current efficiency was found to be exceedingly low in comparison with that obtained with more dilute or more concentrated hydroxide. The suggestion naturally occurred to start electrolysis in strong sodium hydroxide solution, electrolyze until the free base was about 4 percent, then carbonate by passing in CO_2 gas, and finish the electrolysis. But this method increases the complication without possessing any marked advantages. The initial anolyte which gives the maximum efficiency and yields pure sodium permanganate free from manganate, is a solution of sodium carbonate. During the electrolysis with this anolyte, sodium is transferred out of the anode compartment, carbon dioxide is evolved, and there is built up a concentration of permanganate. These changes do not, however, affect the efficiency or voltage until fairly high concentrations of permanganate are reached. It is possible to carry on the electrolysis until all the carbonate is removed, if desired.

Current Density: The current density has a very marked effect upon the yields obtained. The current efficiency is low at low current densities, rises rapidly at first with increasing current density, then less rapidly after a current density of about 60 to 80 amperes per square foot (7-9 A. per sq. dm.) has been reached. The change of voltage with current density is along practically a straight line, the higher current density giving, of course, the higher voltage. With these relations of current efficiency and voltage, the energy consumption curve possesses a minimum which occurs at a current density of 60 amperes per square foot (7

A./sq. dm.). The curve rises but slowly up to 120 amperes per square foot (13 A./sq. dm.).

Temperature: The current efficiency decreases rather rapidly as the temperature increases. The voltage also falls off with rising temperature. As a result, the curve of energy consumption *vs.* temperature possesses a minimum, although not such a sharp one as in the case of current density. The minimum energy consumption occurs at a temperature of about 30° C., although there is but slight variation in energy consumption between 20° and 30°.

DEVELOPMENT OF PROCESS IN LABORATORY.

While ferro-manganese anodes at the start work very smoothly and continue to do so for about twenty-four to thirty-six hours, there comes a time when the voltage begins to rise rather rapidly. This rise in voltage is attended by a diminished yield and the development of considerable heat. Examination of the anode surface shows it to have become coated with a firmly adherent skin or coating. High temperature hastens the formation of this skin, and the amount of the deposit increases rapidly if the anodes are not removed when the voltage begins to rise.

Investigation showed definitely that the rise in voltage was due to the formation of this insulating skin, and not to any passivity effect. The skin consists primarily of iron and manganese oxides which apparently are deposited slowly at all times, but with increasing rapidity as soon as the exposed surface begins to be decreased enough to cause local heating.

The most obvious way to attack this problem was to try to prevent the formation of the skin by the use of various addition agents. The nature of the product, of course, eliminated practically all organic substances from consideration. A considerable number of other materials were tried without success. The effect of anode composition on the skin formation was not thoroughly studied, although it was found that the higher the iron content of the ferro-manganese the greater the difficulty with the skin effect.

In the absence, then, of a means of prevention of this skin effect, several methods were tried for removing the coating after

formation, in order that the electrolysis might be continued. The three most important methods tried were: use of the coated electrode as cathode, treatment with hydrochloric acid, and mechanical treatment.

By making anode and cathode both of ferro-manganese, a simple arrangement was devised by which both electrodes were raised simultaneously, swung around, and lowered again, so that, with the wires interchanged, the anode became the cathode and *vice versa.* This reversal would be made as often as the voltage commenced to rise, this rise indicating the skin effect. It was usually possible by thus using the ferro-manganese anode as a cathode to rejuvenate one anode by the time the other became coated, but the method was not absolutely certain.

It was found that the coating could be removed fairly readily with concentrated hydrochloric acid. The method, however, besides being disagreeable on account of the chlorine that is liberated, is expensive because of the quantity of acid as well as ferro-manganese used up. It is impracticable to avoid dissolving a bit of the anode during each treatment.

The simplest and most effective treatment is the removal of the coating mechanically, that is, by grinding or sand-blasting. The use of a grinding wheel is satisfactory in the laboratory, but the better method of the two, especially for large-scale operation, is the use of a sand-blast. The sand-blast removes the coating completely down to the metal surface, but does not cut into the latter. After continued alternate sand-blasting and electrolysis the anode surface still remains smooth, and experience showed that the treatment may be repeated indefinitely with perfect restoration of original electrolyzing conditions after each sand-blasting.

DESIGN OF CELLS.

Various cells were designed and tried out in the laboratory. The materials which enter the construction of the cells must be unaffected by alkali and by permanganate. For containing vessels concrete and iron were tried. Concrete is reasonably satisfactory provided that a lining of paraffin or similar material, inert chemically, is put in to prevent absorption of the solution by the concrete. Iron, however, was found to be the better material,

since it is inexpensive, non-absorptive, and only very slowly affected by either alkali or permanganate. Tests extending over several days were carried out with a two-diaphragm cell constructed of thin sheet iron. The two side walls of the cell served as the cathode. The cell had three compartments—two outer (cathode) compartments, and one central (anode) compartment. The latter contained two anodes, one placed close to each of the two diaphragms.

The diaphragms used in this cell were made of several sheets of thin, long-fibred asbestos paper supported on each side by narrow strips of iron placed about an inch apart. A number of materials were tried, but asbestos was found to be the best, everything considered.

The current efficiency obtained in the sheet-iron cell with asbestos diaphragms was equal to that obtained in small glass cells and slightly better than in the large concrete cells.

Several experimental runs were made in the laboratory using cells which had an output of about a half pound (225 grams) of sodium permanganate in twenty-four hours. In some cases the concrete cells were used, in others the sheet-iron cell just described. The object of these experiments was to simulate large-scale operation. Both the continuous or cascade and the intermittent or "batch" systems of operation were tried. Both, it should be mentioned, may be made continuous in the sense that the current may be kept passing through the cells constantly.

In the cascade system several cells were arranged in a row on a slight incline in the manner customary where the electrolyte is circulated. The anolyte was caused to flow from the anode compartment of the first cell to the anode compartment of the next and so on throughout the row. Pure sodium carbonate was fed into the anode compartment of the first cell, and permanganate solution of about 12 percent concentration issued from the last. Rather careful adjustment of the initial concentration and the rate of flow of the carbonate solution is necessary in order to insure that the final product will be of the proper composition. A disadvantage of this system is that a cooling coil in each cell is required.

The batch system, in its simplest form, consists in filling the anode compartment of a cell with carbonate solution and elec-

trolyzing until the permanganate has reached the desired concentration. The anolyte is then drawn off and another batch started with fresh carbonate solution. On a technical scale, the scheme would be extended to include a number of cells and a tank with circulation of the anolyte from the tank to the cells (in parallel) and back again. The same anolyte would be pumped through the cells over and over again until it had the desired composition. Cooling would be accomplished very simply by means of one large coil placed in the tank.

Based on the laboratory experiments, a tentative design for a plant was drawn up and a *preliminary* cost estimate made. Based on an output of one thousand pounds (455 kg.) per day in the form of a ten percent solution, it was estimated that under war conditions sodium permanganate could be made for eighty or ninety cents per pound. (See final estimate, p. 384.)

DEVELOPMENT OF PROCESS IN COMMERCIAL CELLS.

The promise of the laboratory experiments showed the great desirability of a trial of the process with cells that could actually be used in commercial scale production. Such a trial would furnish the data for a more exact plant design and a closer estimate of the cost of production. It might also uncover some unforeseen difficulty in large scale operation.

After some search, there was found in use at the plant of the National Electrolytic Company at Niagara Falls a cell which could be readily modified to suit the needs of this process as previously outlined. In a fine spirit of loyalty, this company gave the use of one of its buildings for experimental purposes, and, under the auspices of the Gas Defense Division, a semi-plant scale investigation was started, with the idea of expanding to a production basis at the earliest possible moment.

Description of Apparatus: It is not the purpose of this paper to go into detail concerning the apparatus used in the semi-plant scale experiments, but a brief description will not be out of place. The following flow sheet indicates in a general way the various items required:

There were four main tanks, a dissolving tank, a filter and two cells. Two of the tanks, with the anode compartments of the

cells, connecting piping, and centrifugal pump comprised the anolyte system. An exactly similar set of tanks, piping and pump with cathode compartments comprised the catholyte system. All tanks and piping were of iron.

Two cells were operated in these experiments. They were made of cast iron, and contained two compartments—anode and cathode—separated by an asbestos diaphragm. The effective diaphragm surface was 32 inches (80 cm.) square. The cathode compartment was entirely enclosed except for a vent to permit the escape of gas, while the anode compartment was open at the

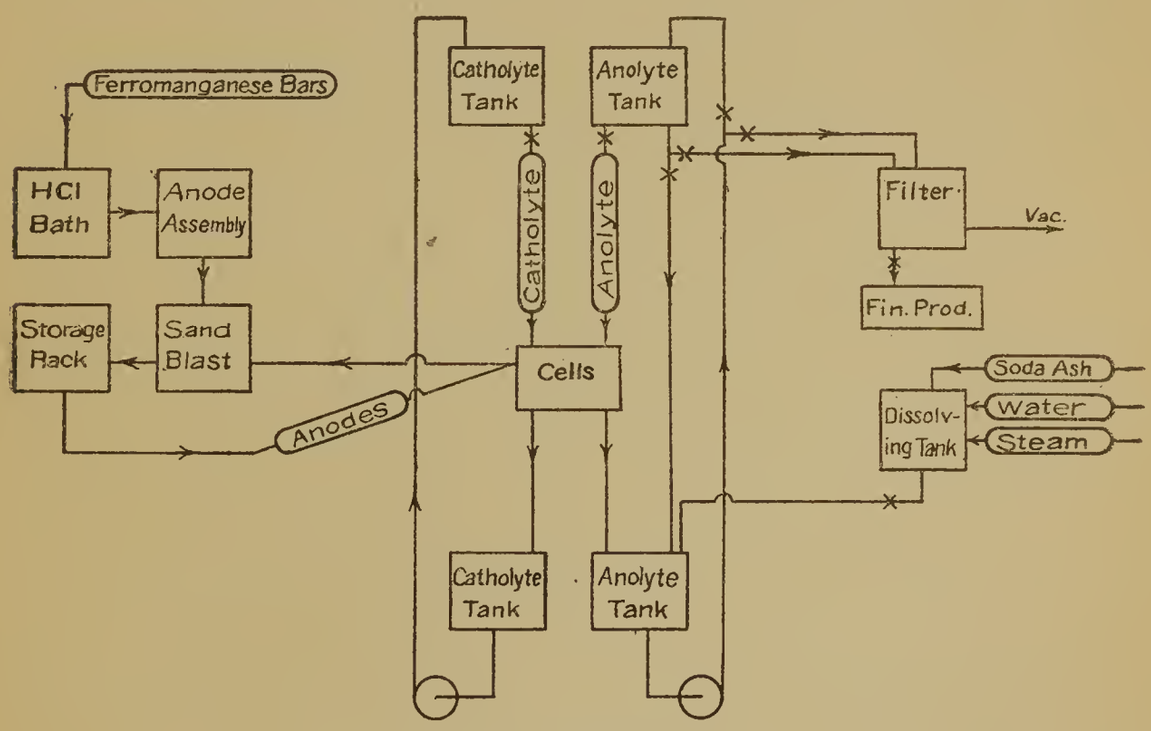

FIG. 1. Flow Sheet.

top to receive the anode. Appropriate piping to and from the cell was arranged to permit the circulation of the electrolytes.

The anode was made up of five bars of ferro-manganese, each having the dimensions 5 by 1½ by 35 inches (13 by 4 by 90 cm.), clamped together near the top between steel rods. To insure uniform contact between the steel and the ferro-manganese, a sheet of lead was interposed on each side. The contact surfaces of both the steel and the anode bars were polished on a grinding wheel before assembling. The steel bars served to lead in the current and contact was made at both ends with cable terminals.

All the anode bars were shipped from New York to Niagara

Falls, packed with excelsior in individual boxes. Out of twenty-eight bars shipped, only one was lost through breakage in transportation.

Method of Operation: The experiments were carried out according to the batch system previously described. The method of operation as finally adopted was as follows:

The anode bars were treated with hydrochloric acid to remove an outer coating formed during casting, after which they were assembled and cleaned with the sand-blast. For each anode in service, a spare anode was kept on a storage rack until needed. Whenever an anode began to display a rising voltage it was reversed, and when the second side likewise became coated the anode was removed from the cell and a spare put in its place.

The catholyte was an approximately eighteen percent solution of sodium hydroxide. It was circulated constantly at a slow rate, water being added from time to time to keep the concentration at the desired value. The initial anolyte was about eighteen percent sodium carbonate solution. The proper amount of this solution was prepared, pumped to the upper tank and allowed to flow by gravity through the cells, overflowing into the tank below. The solution was then pumped back to the upper tank, circulation in this fashion being kept up throughout the run. A run lasted until the batch of anolyte attained the desired composition, namely, about 12 percent $NaMnO_4$ and about 1 percent Na_2CO_3. In these experiments, the length of run varied between seven and twelve days. The conclusion of a run consisted simply in collecting and filtering the whole anolyte to remove the sludge of manganese and iron oxides which is always formed.

Results: The experiments with commercial cells showed the following results:

1. Current efficiency 35 percent, cell voltage 7.0, and energy consumption 12 kilowatt-hours per pound (26.5 kw.h. per kg.) were obtained. This agreed very closely with the results obtained on a small scale.

2. The concentration in the cells should not be carried much above 8 percent $NaMnO_4$. At higher concentrations the current efficiency drops off somewhat and the anodes need changing more frequently.

3. The use of the sand-blast as a means of cleaning anodes when they become coated was given a thorough test and proved entirely adequate.

4. Runs at a current density of 120 amperes per square foot (13 A. per sq. dm.), which is twice that at which minimum energy consumption occurs, demonstrated that the anodes behave satisfactorily and that the cells may easily be kept cool at this high current density. The high yield per cell per day more than offsets the slight increase in cost due to increased energy consumption.

5. It is preferable and entirely practicable to operate at an average temperature of about 20° C.

6. The process as evolved was demonstrated to be entirely practicable for large-scale operation, and there was no new development to indicate any great change in the estimated cost of operation.

RECOMMENDED PROCESS FOR HALF-TON PLANT.

All things considered, the batch system of operation as previously described has shown itself to be the one best suited to the electrolytic production of sodium permanganate. Among the chief factors that lead to this conclusion are the simplification resulting from doing the cooling in one place outside of the cells, and the avoidance of the necessity for careful adjustment of the rate of circulation. Furthermore, uniformity of operation is gained by having all the cells throughout the plant operating on electrolyte of the same composition. With the batch system, the process will be intermittent with respect to the product turned out, but continuous in the sense that the current will be kept passing constantly through the cells, which will never be emptied of electrolyte except for purpose of cleaning and of renewal of the diaphragms.

The interval of time selected for turning out of a single batch is one week. This interval is fixed by the amount of solution placed in the anolyte system and the final concentration of permanganate to be attained in the cells. One week is selected because if the interval is much greater the tankage required becomes excessive. On the other hand, if the interval is much less than one week the "turn-over" becomes too frequent, resulting in in-

creased labor requirements, and the condition is approached of having too small a volume of liquor to permit enough being outside the cells for adequate cooling and for settling out of the sludge.

The concentration of sodium permanganate to be attained in the cells is limited to about eight percent, for the reasons previously pointed out. Fortunately, the evaporation of the solution may be carried out easily and inexpensively and if 30 percent solution is to be produced, as was desired, it makes little difference in the final cost whether the evaporation is started from 8 or from 12 percent.

There are four principal operations to be carried out in this process; namely, the preparation of the solution, anode manipulation, filtration, and evaporation. When the plant is started for the first time an eighteen percent caustic soda solution has to be made up. During normal operation, however, there is only the simple preparation of a 12 to 14 percent solution of soda ash, an operation that needs no further discussion.

Anode manipulation is a question which demands more attention than it does in the majority of electrolytic processes. It is planned to have the cells arranged in banks of approximately eight each. It will then be possible to handle the anodes from all of the cells in one bank, simultaneously, using a small overhead crane. Thus, after shutting off the current from the bank, eight anodes together will be raised, reversed, and lowered or carried to the sand-blast room to be cleaned, another set being put in their place. The sand-blasting can be effected while the anodes remain suspended from the trolley which conveyed them to the sand-blast room. The semi-plant scale experiments showed that the anodes would run more than twenty-four hours before needing reversing, and a slightly less time on the second side before removal altogether. It is therefore planned to make operations uniform by reversing the anodes in one-half of the plant each twenty-four hours, and to carry on replacement in the other half during the corresponding interval. This work would not, of course, consume the whole twenty-four hours, but could readily be carried out by two men during an eight-hour shift.

In this process a sludge consisting of a mixture of the oxides of iron, manganese and silicon is always formed. It is planned

to remove most of this by settling in the upper anolyte tank. For the purpose of washing the sludge and filtering the final product vacuum filters are installed.

The Government desired to ship and use a solution containing thirty percent of sodium permanganate in which the presence of three or four percent sodium carbonate was entirely permissible. The filtered product from the cells therefore must be concentrated. This may be brought about in open tank evaporators containing steam coils to raise the temperature of the solution to about 60° C. and having provision for the continual agitation of the solution by bubbling air up through it. A small loss (about three percent) occurs during the evaporation due to decomposition with the formation of manganese dioxide. Consequently a final filtration should be made before the finished product is ready for shipment. The thirty percent solution may be stored in steel tanks and shipped in steel tank cars.

The power requirements and arrangement of circuits will be mentioned briefly. All the experiments which have been made are consistent with the belief that the energy consumption will be not greater than twelve kilowatt-hours per pound (26.5 kw.h. per kg.) of sodium permanganate produced. Allowing a loss of three percent in evaporation, and, say, ten percent due to inactivity of a portion of the plant, a producing capacity in the cells, of 1,130 lb. (514 kg.) per twenty-four hours must be allowed in order to turn out a thousand pounds (453 kg.) of the finished product. The power required is therefore found to be about 570 kilowatts. It is planned to generate this power at about 112 volts. The cells will be connected in series-parallel, with a third wire running between in such a way that a single bank of eight cells may be cut out with very little disturbing effect on the current passing through the remaining cells. Running at a current density of 120 amperes per square foot, each cell exhibits a resultant drop in potential of about 7 volts.

CONCLUSION.

In conclusion it should be stated that the signing of the armistice caused the discontinuance of the work before the full program of large-scale runs was completed, and actual production was therefore not undertaken. The process does, however,

appear to be very promising as a war-time proposition when the demand for pure permanganate is urgent and the price that can be paid much greater than in peace times. It is practically certain that, had the war continued, production would have actually commenced within a short time.

A design has been made and submitted to the Government giving rather detailed plans and specifications for a half-ton plant and including an estimate of the cost based on the best information in the light of all the work done. This estimate assumes the cost of raw materials, labor, and construction to be as they were in the year 1918, and provides for writing off the plant in two years.

Based on these war-time conditions a fairly accurate cost of building and equipment (not including a sub-station, since the power cost estimate is based on 110-volt D. C., delivered to the plant), for such a plant is $85,000. Pure sodium permanganate in 30 percent solution could be produced for about 60 cents per pound. This cost can be further analyzed as follows:

Raw materials	$0.183
Power at $30 per H. P. year	0.067
Amortization	0.120
Labor and overhead	0.090
Contingencies	0.140
Total	$0.600

While a thorough study has not been made of the production of potassium permanganate, the evidence thus far obtained indicates that solid $KMnO_4$ could be produced somewhat more cheaply than the sodium salt. Both products could, of course, be made much more cheaply if labor and materials again became reasonably normal.

It is planned to continue the further investigation of the production of potassium permanganate in the Research Laboratory of Applied Chemistry of the Massachusetts Institute of Technology.

DISCUSSION.

F. A. LIDBURY[1]: I have been reading this paper and listening to the exposition which has been given, and I am struck by the fact that such comparisons are made between this process and the older processes as are made between the concededly abnormal selling prices of sodium permanganate and cost estimates based on the newer method. Now it seems to me that comparisons of that kind do not get one anywhere, and I am not without suspicion that the whole inception of this work was due to a comparison of that kind having been initially made instead of a comparison of the possible cost of sodium permanganate by the older method with its possible cost by this. Therefore I should like to inquire whether any investigation, technical or commercial, was made as to the possible cost of making sodium permanganate by the older and better known method.

W. G. HORSCH: A considerable amount of study of the older fusion process has been made by others in Major Wilson's laboratory and the complications inherent in that method make it appear that the cost would be greater even under the most favorable conditions.

F. A. LIDBURY: Have you any estimate, to compare with the estimate you have given, of the cost by this method?

W. G. HORSCH: No, I have not. Major Wilson may have.

F. A. LIDBURY: It seems to me, at first sight at any rate, quite reasonable to imagine that it ought to be possible to make sodium permanganate very much more cheaply by the older method, involving very much cheaper raw materials and not much greater anolyte cost than by this method, and the paper would be a great deal more valuable if comparisons were made between cost estimates rather than between concededly abnormal *price* value on the one hand and *cost* estimates on the other.

R. E. WILSON: I think attention might be called to one statement in the paper as to the reason we undertook this investigation, that even disregarding the cost basis, the best quality that could

[1] Works Mgr., Oldbury Electrochemical Co., Niagara Falls.

be made by the fusion method was not pure enough for the Government requirements; the best that could be done was to cut the sodium chloride content to 25 percent of the sodium permanganate present in the solution, and that caused serious deterioration in the resulting soda lime. The first object of this work was therefore to make a purer product. The reason we have not given details and cost estimates for the older process is that we have not carried the cost estimates quite so far; we have not designed a plant and figured everything that went into the plant, but before we undertook the large-scale electrolytic investigation, we did study the fusion reaction in the laboratory and went into it very thoroughly. The losses in the fusion are very large, since it is so much more incomplete than the potassium permanganate fusion, both caustic and manganese dioxide being used up without any return. The difficulties in oxidation and concentration are so great that we could not estimate the cost under war conditions at less than $1.00 or $1.10 per pound. That was based, not on detailed plant design, but the best figures we could get outside of that. This was to some extent corroborated by the fact that the Government was in the market for over a year and pestered every manufacturer they could possibly persuade to make sodium permanganate, and no one sold it at less than $1.75 a pound, claiming that their cost was over $1.50 per pound. Their books were verified by the Government. I can, however, see no reason why it should be above $1.10, even under war conditions.

F. A. Lidbury: You speak of the large amount of chloride that occurs as an impurity in permanganate by the fusion method; where does that come from?

R. E. Wilson: From the oxidation process; by the fusion process sodium manganate is made and then leached out. There are three chemical methods of oxidation used; the addition of sulphuric acid, which leaves sulphates as impurities, which are much worse than the chlorides; the use of chlorine, which gives the best product, though even that contains impurities, and the use of CO_2, which gives the purest product but is a very difficult proposition to operate, the addition of the CO_2 requiring days before it is completely oxidized. Furthermore, the two acid methods of oxidation, oxidize the manganate only at the expense

of itself, two molecules going to permanganate and one going to manganese dioxide, which makes a sludge and involves considerable loss while the chlorine method oxidizes all the manganate with practically no loss.

F. A. LIDBURY: Well, there is another method, that of oxidizing the manganate electrolytically.

R. E. WILSON: That has also been gone into very thoroughly, and while the oxidation is relatively cheap, if you use manganate liquor as ordinarily obtained from the fusion of sodium and manganese, the solution contains 7 percent sodium manganate and 18 percent of partially carbonated alkali, and to make a stable solution you must not only oxidize the manganate but you must also electrolyze out the caustic. This requires as much power and nearly as much equipment as to make it from the ferromanganese to start with. Since ferromanganese is a much cheaper source of manganese than is sodium manganate, it was figured on the basis of careful estimates that the ferromanganese process had an advantage over the other process of fully 15 cents or 20 cents per pound.

Presented as part of a Symposium on "Released Electrochemical Information" at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

LEAD PLATING OF SHELL INTERIORS AND BOOSTERS.

By A. G. Reeve.[1]

Abstract.

The object of this paper is to give a detailed description of the apparatus and process utilized in the practical application of well-known lead electro-plating formulae in solving two problems in the manufacture of certain lachrymatory gas shells.

First: As some of the liquid lachrymatory gases deteriorate rapidly when in contact with steel, a protective coating is necessary on the inside of shell and the outside of booster, with perfect unbroken junction where the two are screwed together.

Second: Reclaiming underweight rejections in the production of high explosive and gas shells.

GAS SHELL PLATING.

Electro-silver deposits were used in the first experiments, as this also resists the chemical action of the gases, and the equipment was conveniently available. Tests indicated the desirability of thick deposits, and lead was resorted to as being equally rapid, fully as effective, and less expensive.

To properly protect the gas from contact with the steel, the lead deposit must adhere to the inside of shell, be continuous, and free from pin-holes or porosity. Maximum perfection in each of these requirements can only be attained by suitable preparation of the surface to be plated, as in all ordinary plating, this being somewhat more difficult owing to inaccessibility of the surface being treated.

[1] Plating Adviser, Oneida Community, Ltd., Oneida, N. Y.

After removing oil, if any, with gasoline, the shells were filled with a hot alkaline electric-cleaner solution, connected as cathode, and with anode inserted in nose of shell, 6-volt continuous current producing copious gas evolution for about 30 seconds removed greasiness.

The 4.7 inch and 155 mm. shells having had the scale removed by sand-blast prior to the above operation, were rinsed and pickled electrically (as cathode) with a cold solution of one part commercial HCl to five parts water, and after rinsing this out

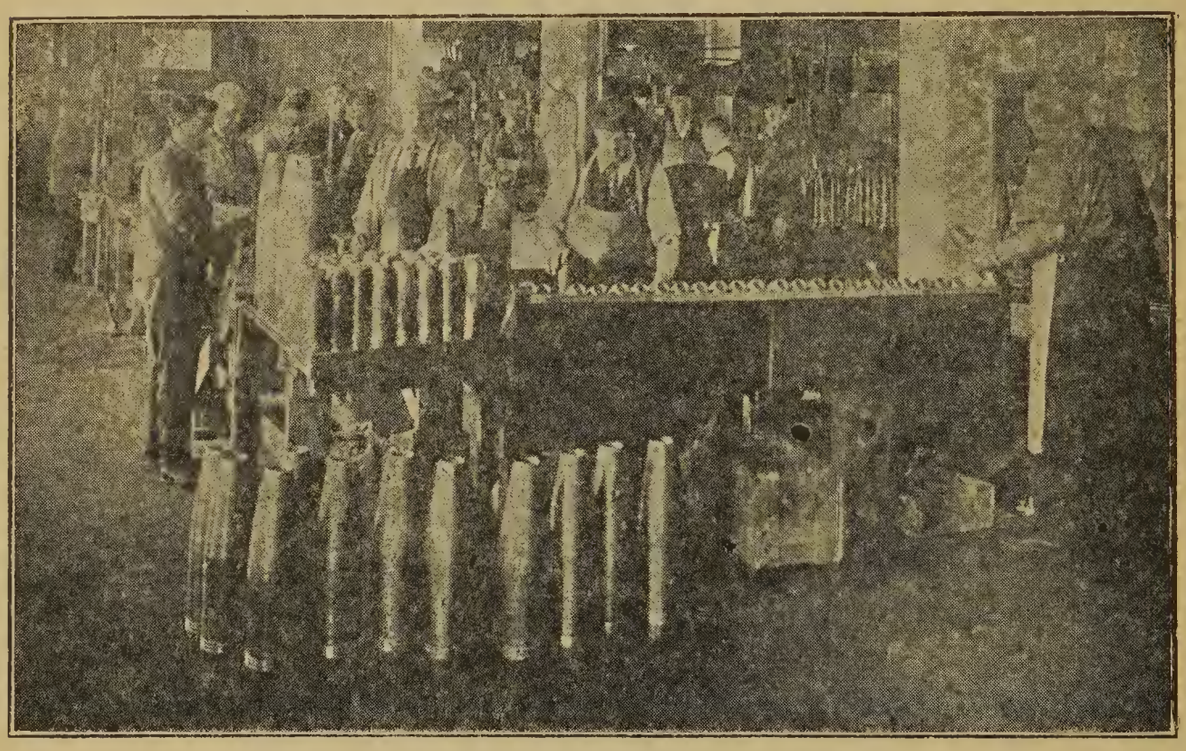

Fig. 1. Shell Plating—Inspecting and Testing.

were quickly filled with hot copper "striking" solution and given a preliminary coat of copper. The 75 mm. shells that were not sand-blasted had their scale removed by a revolving steel-wire brush with crushed flint and water, immediately following the electric pickle, and were then "struck" with hot copper bath same as above, made up as follows:

Basic copper carbonate	8 oz.	(225 grams)
Sodium cyanide	34 oz.	(960 grams)
Water	1 gal.	(3.8 liters)

Analyzing about 4 oz. (30 g.) copper and 2½ oz. (20 g.) free cyanide per gallon (liter). Lead deposits from the fluosilicate

bath were not permanently adherent when made direct upon steel, hence the copper "strike."

LEAD BATH.

The fluoborate and fluosilicate baths were both used, chiefly the latter, however, because of the ease with which it could be procured from electro-lead refineries. The fluoborate bath will deposit a smooth adherent plating direct upon steel without an intervening coat of copper, and for all ordinary work this would seem to be a decided advantage; but in this case, however, the copper "strike" and subsequent inspection made the process more fool-proof and shells having small particles of scale still adhering were much less likely to be given the lead deposit prematurely, than if inspected without the copper strike-coating. The character of the "strike" deposit is very important, and after rinsing, is inspected inside with a bright electric auto-lamp. If the "strike" deposit was imperfect, the cause was removed and the operation repeated. Too heavy "strike" or improper preparation of the underlying surface will cause a rough, porous or powdery copper deposit, which in turn induces porosity and "treeing" in the subsequent lead plating. The copper anode was not rotated, and owing to the great excess of cathode over anode surface, the strike bath became rapidly impoverished and chemical control was very essential, with frequent tests and additions.

ANODE AND STIRRER FOR LEAD BATH.

The cathode surface is over five times that of the anode, and the quantity of bath is small in proportion to the volume of metal to be deposited. Thin, flat anodes were cast with the widest part near the center and only one-half inch (13 mm.) wide at top where they come through the nose of the shell.

The stirrer was a thin maple paddle as wide as would go through the shell opening. The lower end of the anode passed through a transverse slot near the bottom of the paddle, was bent downward around the end and upward on the opposite side, terminating about one-quarter inch (6 mm.) above the bottom end of the paddle. The top portion or shank of the stirrer paddle (⅝ inch (16 mm.) square by 4 inches (10 cm.) long) was deeply

recessed to receive the upper narrow connecting end of the anode and also the stirrer spindle from which it received current.

The anode and stirrer arrangement was the subject of much experimentation. This scheme of holding the anode flatwise against the stirrer-paddle considerably minimized the active anode area, but the reduction of surface by running the slender strip through the recess in the paddle shank, and the increase of surface exposure by four large holes in the central portion of the paddle, together with the return bend of the anode at the lower extremity, equalized the current densities which otherwise would

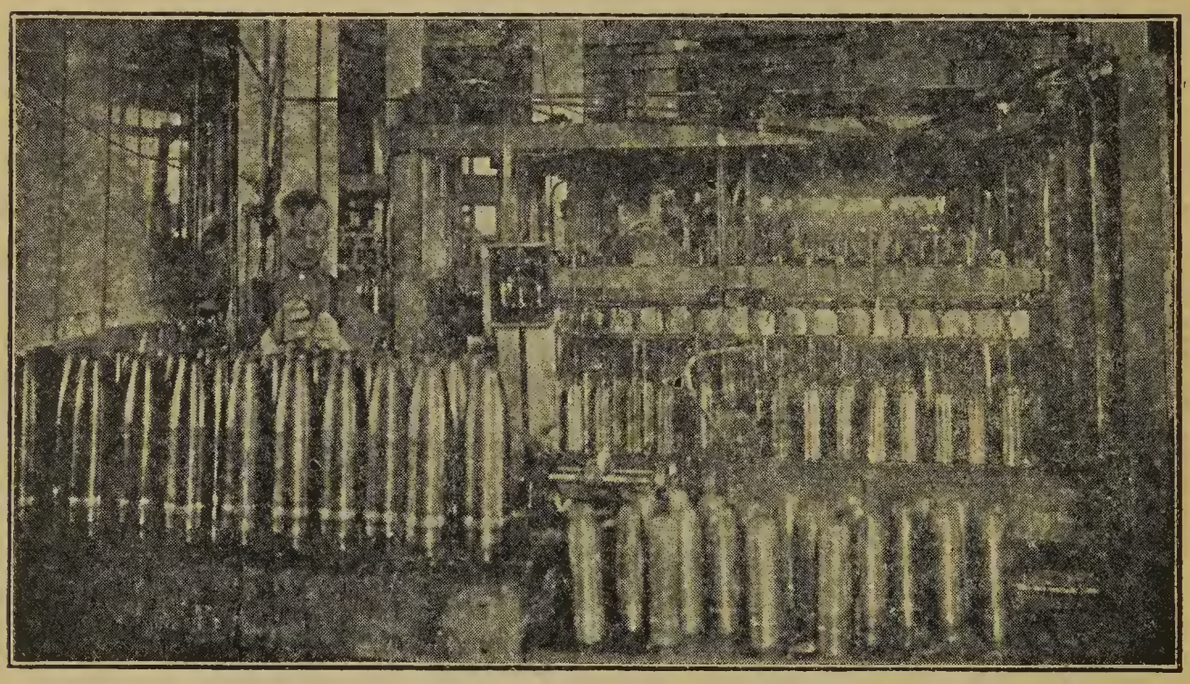

Fig. 2. Shell Plating—Revolving Spindle, Plating Units.

have made the deposit in the threaded opening too rapid and insufficient in the central portion and bottom. Rotating the anode and stirrer paddle (320 R.P.M.) compensated in a measure for insufficient anode area. Owing to the simplicity of both paddles and anodes they were produced with minimum labor and were assembled in quantities by women, in advance ready for use.

LEAD PLATE.

Shells, copper "struck", filled with lead bath, anode and paddle, were placed on the plating bench (fifty-two 75 mm., 4.7 inch or 155 mm. shells per bench unit), anode and paddle quickly attached to the rotating spindles, and deposition begun.

Each spindle could be raised and disengaged from the driving pulley independently. The entire frame carrying the 52 spindles could be readily raised or lowered to suit the various shell sizes.

Current was supplied to the spindle through its two babbitt bearings set in woodwork. To use the available ampere-hours advantageously, the 52 spindles were divided into four batteries and connected in series, with 6-volt plating current, the 13 spindles of each battery being connected in multiple. To insure uniformity each shell was given a new anode, the old ones being re-cast.

The specification for gas shells required that the thickness of plating should not be less than ten nor more than fourteen thousandths inch (0.25-0.35 mm.) for all sizes, except that five thousandths (0.125 mm.) was desired on the threads in the opening. This was accomplished by withdrawing from the shell, when half plated, a small portion of the bath by use of a rubber bulb and a brass tube bent at the end so as to reach down only the required distance into the shell opening. To further equalize the thickness of deposit, each shell was turned part way around on its axis several times during the plating period.

The following table gives approximate data on the volumes, areas, amperes, and time, etc., for three shell sizes:

	Exposed Anode Surface Sq. In.	Cathode Surface Sq. In.	Ounces Lead Deposited	Amperes	Hours
75 mm.....	11.50 (18 sq. cm.)	64 (100 sq. cm.)	4.51 (127 g.)	5	6.63
4.7 inch.....	20.07 (31 sq. cm.)	147 (229 sq. cm.)	10.21 (289 g.)	10	7.50
155 mm.....	37.06 (58 sq. cm.)	266 (412 sq. cm.)	18.61 (529 g.)	20	6.84

The bath as received from the refiners ready for use analyzed about as follows:

Deg. Baumé	Percent Pb	Percent H_2SiF_6	Percent Free H_2SiF_6	Percent Glue
43	18.02	14.14	2.90	0.4

This was supposed to work best at 95° to 100° F. (35°-38° C.) with from two to five percent free acid, and in practice it was found that the resistance of the bath was such that with 6 volts the shells and bath soon warmed up to the desired temperature while the plating was going on.

Regardless of the recommendations of those who furnished the lead bath, the quality of the deposit was materially improved by the addition of 2 lbs. of glue to each barrel (50 gals.) (5 g. per liter) of new bath before using. As fast as the plating operation was completed the bath was poured out of the shells and filtered, and one lb. of glue added per barrel of bath (2.5 g. per liter) each time it plated a set-up of shells. The number of shells set up for plating with one barrel of bath and the volume of lead deposited per barrel set up, are given below:

	75 mm	4.7 inch	155 mm
No. of shells filled per barrel (190 liters) of bath....	346	100	42.37
Lb. of lead deposited per barrel (190 liters) of bath..	97.3 (44.2 kg)	64 (29.1 kg)	49.3 (22.4 kg)

It will thus be noted that in the 75 mm. shells each gallon of bath deposited nearly 2 lbs. of lead in 6.63 hours, with one glue addition. The discoloration which the exterior of the shells acquired was scoured off by a revolving brush with an abrasive and water. After rinsing in hot water (made alkaline with sodium carbonate) they were dried and inspected.

TESTING.

The thickness of plating was judged by dropping a ⅜ inch (9 mm.) steel ball through a perpendicular six-foot (1.8 m.) glass tube, the lower end of which rested upon the bottom inside of shell. The rebound was very slight when the plate was of the required thickness. More difficulty was experienced in getting the lead to deposit on the bottom than elsewhere, so that if the bottom indicated well it was reasonably certain that the balance of the interior had the desired thickness. To detect pin-holes or porosity, Dr. Blum, of the Bureau of Standards, recommended the Ferroxyl Test applied as follows:

First, wash out the shells for three minutes with a 5 percent solution of H_2SO_4 to remove alkalinity and traces or particles of iron that may have become enmeshed in the lead deposit. Second, fill the shells for one minute with the Ferroxyl solution (2 percent K_3FeCN_6 + 2 percent H_2SO_4). The deep blue color will clearly indicate steel not properly covered.

BOOSTERS.

The cylindrical shape of the boosters enables them to be plated as rotating cathodes on the shell-plating spindles at 400 R.P.M., thereby obtaining commercially, in a measure, the ideal deposits such as are often attained in laboratory practice. At first some difficulty was experienced in obtaining sufficient deposit in the apex of the right angle formed at the junction of the booster proper with the threaded adapter, but the difficulty was soon overcome by adding about 10 lb. glue per barrel (24 g. per liter) to the lead bath every 24 hours. A good grade of nitrogenous glue was used for all additions.

WEIGHTING LIGHT REJECTED SHELLS.

This was confined entirely to 4.7 inch shells and larger sizes. The process was similar to the lining of gas shells except that the majority of underweight rejections were high-explosive shells, and these had a preliminary coat of silver substituted for the copper "strike" deposit. A non-porous plating was not as essential in the high-explosive shells, and the weight of lead required varied from 2 to 14 ounces (57 to 400 g.) per shell.

DISCUSSION.

Wm. Blum[1]: I think this report is a very interesting illustration of the development of an industry, not great in itself, to meet war needs. Prior to this demand for lead plating, there were only a few plants in the United States that ever conducted

[1] Chemist, Bureau of Standards, Washington, D. C.

lead plating on a commercial scale, chiefly in connection with storage battery fittings. Within a couple of months of the time when this demand for lead plating arose, there were 20 or 25 plants in the United States, and particularly the Oneida Community, with which Mr. Reeve is connected, which has made very great progress in that industry. Among new possibilities for commercial lead plating is the application to the chemical industries, since lead is very frequently desired as a lining for chemical equipment. All of the work which was done in connection with the ordinary gas shells and boosters was with comparatively light coatings of lead. They were heavy for plated coatings (0.010 inch (0.254 mm.) of lead is a heavy electroplate as compared with electroplates of nickel or copper or other metals usually plated) but that was by no means the limit. It was found entirely possible on a commercial scale to deposit lead up to 0.10 inch (2.54 mm.) in thickness, which was quite impervious and resistant, and adherent to the surface of the steel. In October and November, 1918, under the direction of the Bureau of Standards, there was plated for the Edgewood Arsenal, one of the large catalyzer boxes to which reference was made this morning. On an inside surface of over 50 square feet (4.6 sq. m.) there was plated 275 pounds (125 kg.) of lead; which represents an average thickness of about 0.07 inch (1.8 mm.). This is of interest to chemical manufacturers who may desire to have lead coatings on chemical apparatus, which can be produced conveniently by the present method. Unfortunately, there was only one tank plated at the time the armistice was signed, and that never went into service, but there is every reason to believe that lead plating is a perfectly feasible commercial operation for any industry which desires or requires lead lining for steel.

Linn Bradley[2]: It may be interesting as a matter of record to state that I was engaged in a little experimental work on the lead plating of a gas shell in December and January, a year ago, December, 1918, and we had the same difficulty recorded this afternoon in plating the bottom of a shell, and we eliminated that difficulty by circulating our electrolytes; instead of having a rotating anode, we simply hung our quiet, still anode in a shell

[2] Chief Eng., Research Corporation, New York City.

and circulated the electrolytes, and that gave us a very even deposit all the way around the bottom, and in addition, we could run our current density with a proper circulation up as high as 60 amperes to the square foot (6.6 amp. per sq. dm.).

F. C. FRARY[3]: There was one point in connection with that little plating job that Mr. Blum forgot to mention; the first samples he brought down to us at Edgewood looked very nice and we thought they were going to be pretty good till we took them out and put them on the catalyzers. The catalyzers ran pretty hot, considerably over the boiling point of water, and after they had been lying on the catalyzers five minutes, they had the nicest lot of blisters you ever saw. If we took a knife and punctured a blister, there seemed to be lead under and over the blister. Mr. Blum went back to the Bureau and we did not hear any more of lead plating for a while, then he came and showed us some samples that did not do that. The whole trouble was caused by the pickling of the original iron. If the iron was cleaned by pickling, you invariably got blisters; if it was cleaned by sand blasting, you did not get any blisters. It reminded me of the discussion yesterday about those little holes; apparently if you had pickling, you got water or something in them that caused the blisters. If anybody wants to do that kind of a job again, I would warn them to have their machinery sand blasted and not pickled.

E. C. WALKER[4]: We had that same trouble with blistering and traced it to acid in the pores of the metal, but we eliminated that by a caustic dip and have not had any trouble since; we have had samples in boiling sulphuric acid at 125 degrees C., with a condenser over the top of the vessel so that the water would keep going back in. We have had samples of steel coated with ten thousandths of an inch (0.0001 inch = 0.00254 mm.) of lead on test for two months under these conditions, without any blistering whatever.

LINN BRADLEY: I would like to ask the author if he thinks this method would be applicable to making lead-lined iron pipe for chemical and other industries.

[3] Director of Research, Aluminum Co. of America, New Kensington, Pa.
[4] Genessee Chemical Co., Batavia, N. Y.

A. G. Reeve: It would depend a good deal on the size of the pipe. If the pipe was large enough to stir the electrolytes, I do not see why it could not be done; if the pipe was long and very slender, it would be difficult to work out.

Linn Bradley: Would the coating be sufficiently dense that sulphuric acid would not get in and attack the iron?

A. G. Reeve: I think it would. I think it could be done that way. The preparation of the surface, as Doctor Frary states there, is quite important, and the early deposit, the first "strike" coating, as the platers call it, would be very important towards securing a perfectly homogeneous and solid deposit of lead, which would resist the acid subsequently.

Linn Bradley: I may be able to give a little information on that. We plated a solid conductor for electric light purposes, we coated it with ten thousandths of an inch of lead a year ago last February, and they have been in moistened cinders which were kept warm, and in acetic acid constantly since that time, and there has been no indication of any corrosion of the steel—of course that is on the outside. There is a chemical company in the city of New York that has been using lead-coated pails, inside and outside, since the 3d of December, and I saw them yesterday and they were still perfect and they had been handling 30 percent of sulphuric acid in the pails.

G. A. Roush[5]: Mr. Reeve's discussion of lead plating on steel is entirely from the standpoint of the protection of the steel from the gas in the shell. There has also been a limited use of lead plating on the interior of common steel shell for the purpose of bringing an underweight shell up to standard weight. In machining the shell, it is impossible to avoid over-machining in a certain small percentage of cases, thus getting the weight of the shell below the minimum requirement. In this case such shells were sent to the plating bath in the same manner that Mr. Reeve describes, and a thin coating of lead was applied to the inside of the shell in order to bring the weight up within the required limits. In a similar manner, purely as a hospital treatment, as we call it, for bringing up to within the specified limits an over-

[5] Asst. Prof. of Metallurgy, Lehigh University.

machined shell, copper plating has been applied to the outside of the base of the shell behind the copper rotating band, in order to bring up a shell which was of too small a size, thus increasing the diameter of the base of the shell to a sufficient size to make it fit properly into the end of the cartridge case.

GEO. B. HOGABOOM[6]: It may be well to state why boosters were plated as long as Mr. Roush has stated why shells were plated. In the butt-welding of the booster base on to the adapter there was some thought that there might be an oxide of iron formed, and that would be attacked by the phosgene gas. Lead coatings are not attacked by the gas. Also, in the adapter, the lead coating on the threads acted as a gasket and that was found to be quite valuable. If you have unions and want a good, tight joint, it would be well to plate lead on the threads and then put on the caps of your unions, it forms a good strong gasket.

It might be amusing to you to tell of one experience that we had about plating these boosters. The bottoms of one lot of boosters plated by one concern were very rough, and upon asking for an explanation, one of the men stated that ions carried lead from the cathode to the anode, and he said "you know they travel very rapidly, and when they come to the curved surface at the bottom, they slide."

WILLIAM THUM[7] (*Communicated*): I think Mr. Reeve should be congratulated on his application of lead plating to such important work as the production of gas shells. Conditions for this work were very favorable to lead plating. The maintenance of a comparatively high free acid in the electrolyte is essential and with the ratio of cathode to anode surface any deficiency in free acid at the start of the electrolysis was readily corrected within a short time. The small volume of the electrolyte also allowed a quick heating up to the best temperature for plating and the importance of free acid and temperature must not be under-estimated. The importance of temperature is brought out in the paper by Prof. Oliver P. Watts. Mr. Reeve also correctly calls attention to the necessity of glue additions, which necessity is well known to those who have practiced this lead plating art for years. No doubt,

[6] Electroplating Adviser, U. S. Bureau of Standards, Washington, D. C.

[7] Supt., Lead Refinery, U. S. Metals Refining Co., East Chicago, Ind. (Died June 28, 1919.)

the fact that a perfect copper striking was used previous to lead plating made the lead plating all that could be desired as to homogeneity and adherence of deposit, although these same properties may be obtained by directly plating on steel and iron surfaces with the lead fluosilicate as well as the lead fluoborate electrolytes, especially due to advances which have been made in the art of lead plating.

Presented as part of a Symposium on "Released Electrochemical Information" at the Thirty-fifth General Meeting of the American Electrochemical Society, in New York City, April 4, 1919, President Tone in the Chair.

THE PRODUCTION OF SILICON-MANGANESE IN ELECTRIC FURNACES

By B. G. Klugh.[1]

This paper is not intended as an exhaustive or authoritative treatise upon the subject, but merely to present some observations and data obtained in actual practice producing alloys of silicon and manganese in electric furnaces. It is furthermore not confined to the alloy denoted in a strict nomenclature as silico-manganese, in contradistinction to silico-spiegel, mangano-silicon, or other terms, but broadly to such alloys as contain 20 percent or over of silicon and 25 percent or over of manganese.

There was not during the period covered in these observations any definite demand for any specific grades of the silicon and manganese alloys, nor were there always available the most desirable raw materials from which to produce the alloys required. The wire range of the constituents of the silicon-manganese alloys produced and the unusual class of materials treated were governed largely by war-time conditions, or orders demanding special alloys, or by the necessity of converting the raw materials available into those products to whose manufacture they were best adapted. The facts obtained in this experience are presented for whatever value they perchance may have to those interested.

FURNACE EQUIPMENT.

The furnaces used in this work were of the rectangular open-top type. Two sizes of furnaces were used in the principal portion of this work. One furnace, which we will designate in this paper as the *A* furnace, is constructed of a steel shell 15 x 7½ x 5 ft. (4.5 x 2.25 x 1.5 m.) high. It is of a 3-phase type, with 3 electrodes in line. The lining is of carbon with walls 18 inches

[1] Southern Manganese Corporation, Anniston, Ala.

(45 cm.) thick and bottom 2 feet (60 cm.) thick. The tap hole is in the center at one side in line with the middle electrode. Water-cooled copper electrode holders are used. The current is furnished through three 500 KV.A. transformers, being delivered at 22,000 volts on the primary side and reduced to 65 volts on the secondary side. The control of the furnace is effected by means of winch control of the electrodes.

The other furnace, designated herein as furnace *B,* is of the same general type of construction but has a shell 18 x 9 x 5 ft. (5.4 x 2.7 x 1.5 m.) deep, and is lined with carbon with 18 in. (45 cm.) wall and 30 in. (75 cm.) bottom. The current is furnished through three 1,000 KV.A. transformers, being delivered at 22,000 V. on the primary and transformed down to 100 V. on the secondary. The control and tapping arrangement is practically the same as that of furnace *A.*

RAW MATERIAL.

Considerable reference is to be found in the literature upon the theoretical production of silicon-manganese alloys from definite minerals, such as rhodonite, pyrolusite and quartz, or other pure materials. However, no such materials were available, in cost or quality commensurate with economic production, during the period in which these data were taken.

The use of a fairly pure manganese ore, with silica rock, as raw material for silicon-manganese, is hardly commercial, due to the better adaptability of such manganese ores to the production of ferromanganese. The high current consumption in the production of silicon-manganese renders the use of low-cost raw materials imperative.

PRACTICAL DATA.

The operation of above described furnace *A* was recorded over a period covering the production of about 50 tons of alloy of the average analysis of about:

Manganese	65	percent
Silicon	21	"
Iron	13	"
Carbon	below 1	"

The analyses of the raw materials charged were as follows:

	Mn	Fe	SiO_2	Al_2O_3	CaO	MgO	BaO	H_2O
Ore.....	33.11	2.41	15.19	3.96	1.96	0.62	0.50	10.72
Ore.....	34.00	3.19	32.85	3.12	0.00	0.26	0.62	1.00
Ore.....	17.87	2.96	32.97	8.72	0.39	0.50	1.10	22.20
Limestone.....	....	....	0.90	0.42	54.87	0.75	....	....

The above materials were the sources of metal and flux. The reducing agent used was an Alabama bituminous coal containing: Fe: 0.85, SiO_2: 10.00, Al_2O_3: 8.15, fixed carbon: 54.00, volatile carbonaceous matter: 21.00. The metal produced was fairly uniform, of the analysis given above. The slag produced, as compared with that of the calculated chemical balance, is shown in the following table:

	Mn	SiO_2	Al_2O_3	CaO	MgO	BaO
Actual	5.21	36.34	15.28	37.04	1.71	2.39
Theoretical	[5.21]	34.55	16.76	39.02	1.43	1.47

The manganese found in the actual slag is used in the calculation of the theoretical slag, to bring them to a comparative basis.

The slag was at all times free flowing, and with no tendency toward "gumming up" of the tap hole, except where an occasional excess of carbon produced a small percentage of carbides. This is usually incident to the tendency of electrodes to rise out of the charge, with the formation of bridges between the electrodes. Very careful operating control is necessary to preserve the uniformity of operation required for the successful continuous production of metal above described.

The high slag volume, while consuming considerable current, preserved a balance of heat in the high-temperature zone of the furnace. The recapitulation of the data obtained from careful records of the above described run is as follows:

Ratio of ore to metal	3.53
Percent of manganese charged found in metallic alloy	73.00
Percent of manganese charged found in slag	10.02
Percent of manganese charged in gas (by diff.)	16.98
Percent of total weight of charge passing in metallic alloy	16.11
Percent of total weight of charge passing in slag	31.34
Percent of total weight of charge passing in gas	52.55
Pounds slag per ton metal	4,358 = 2,579 kg./ton
Pounds gas per ton metal	7,308 = 3,654 kg./ton
Pounds electrode per ton metal	168 = 84 kg./ton
Pounds reducing carbon per ton metal	1,715 = 858 kg./ton
Kw. hours, per ton metal	9,000 = 10,000 per ton

From the above data it will be seen that the practice is far from ideal economic operating conditions. However, as results obtained with supplies obtainable under war-time conditions, they are submitted, as obtained, for whatever interest they may deserve.

The high slag volume was maintained in order to preserve the alumina balance required for a fluid slag under these conditions. There appears to be a range of slags in electric furnace practice, with alumina increasing from 15 to 30 percent, within which the viscosity increases, producing a sluggish slag which is difficult to tap and tends to keep in suspension large quantities of hot metal.

Use of Manganiferous Slags.

During the early part of 1918, when the shortage of manganese ore was so acute, it became necessary to resort to an accumulation of slag of high manganese content in order to continue operations and supply the demand for manganese. The high manganese slag from all the previous production of ferromanganese in electric furnaces had been stocked, in view of possible future emergency or the development of a means of utilizing it. Several carloads of slag from a blast furnace producing ferromanganese was also obtained. This slag was used as the sole source of manganese and silicon, in the production of silicon-manganese, in several furnaces of the type above described under *A*. A small quantity of iron turnings was used to supply the necessary iron for the alloy and to furnish a more conductive charge in the furnace. There was a considerable variation in the composition of the slag

treated during this period, which covered considerably over a month, and in which time about 100 tons of metal was produced. An approximate average analysis of the production of metal was:

Manganese	70	percent
Iron	9	"
Silicon	20	"
Carbon	below 1	"

A low-ash bituminous coal was used as reducing agent.

A fair average of the analysis of the slag treated and that produced was:

	Mn	SiO_2	Al_2O_3	CaO	MgO	BaO
Slag treated	20.00	34.00	14.00	25.00	2.00	1.50
Slag produced	1.00	29.12	23.02	42.16	2.73	1.61

The slag produced was 4.7 times the weight of the metal, or 9,400 lb. slag per ton of metal. This would be anything but an inviting proposition for normal economic conditions. However, in consideration of the fact that the manganese in the slag used in this test had already been charged off, and the fact that it was at that time the only source of raw material available to keep the plant in operation, the operation in this case showed a profit even with a current consumption during this period of about 15,000 KW. hours per ton of metal produced (16,500 KW. hr. per metric ton).

The slag produced was of a rather sluggish and sticky nature, tending to form accretions in the furnace. This is no doubt due to the alumina being within the range producing such character of slag.

The maintaining of a regular balanced current input on all the phases was accomplished with considerable difficulty, but with an average degree of success.

The interesting technical feature of this operation is that 96 percent of the manganese in the slag treated was recovered in the metal.

An unusual product was made over a considerable period, during the past year, upon a special order for an alloy of approximately —

Manganese	20 percent
Silicon	50 "
Iron	30 "

As in the former cases cited, the high-grade manganese ore on hand was considered too valuable to be used, with the heavy current cost of the silicon reduction. Furthermore, there existed the moral obligation to convert all possible manganese into ferromanganese, which the steel makers required for war material production. Therefore, the above special silicon-manganese alloy was produced from raw materials which were in no way suitable for conversion into ferromanganese. The materials used were:

	Mn	Fe	SiO_2	Al_2O_3	CaO	MgO	BaO
(A) Ore	23.21	8.55	37.00	5.00	1.36	0.25	1.10
(B) Ore	17.50	3.32	47.27	4.24	0.09	0.83	0.74
Silica Rock	....	0.70	96.00	1.00	1.00	1.00	....
Turnings	....	98.00	....	....	....	....	....
Limestone	....	....	0.68	0.32	54.30	1.24	....

The reducing agent used was a mixture of bituminous coal and coke. It was necessary to add lime to the charge in order to produce a slag of sufficient fluidity. This obviously required additional lime, and retaining some silica unreduced to maintain the balance.

Unfortunately, this period of operations was not furnished with the data such as weighing of slag, etc., in order to present an exact metallurgical balance sheet. However, the operation was throughout made with little practical trouble. Contrary to the usual advice that slag is a source of insurmountable trouble in electric furnaces, where alloys of silicon content above 40 percent are produced, there were practically no serious tapping troubles. The tapping arc was not used once during the period of the run on this metal, and the slag usually flowed freely. Even

the cleaning of the metal from the slag was not a serious problem. An average analysis of the slag for the entire period was:

Manganese	3.42	percent
Silica	48.44	"
Alumina	16.95	"
Lime	22.41	"
Magnesia	2.93	"
Barium Oxide	0.70	"

The data of the complete run are as follows:

Ratio ore to metal	2.31
Percent of total manganese in metallic alloy produced	95.00
Percent of total manganese in slag	1.1
Percent of total manganese in gas	3.9
Percent of total weight of charge as metallic alloy	22.58
Percent of total weight of charge as slag	13.42
Percent of total weight of charge as gas	64.00
Pounds slag per ton metal	1,333 = 667 kg./ton
Pounds gas per ton metal	6,359 = 3,180 kg./ton
Pounds electrode per ton metal	227 = 113 kg./ton
Kw. hours per ton metal	9,907 = 11,000 per m. ton
Voltage at furnace	100

The volatilization loss of the manganese was kept quite low in this run, but that of the silicon was about 20 percent of the total charged.

A number of exhaustive analyses have been made upon silicon-manganese alloys through the range of silicon content from 10 to 55 percent. Calcium, aluminum, and magnesium are practically absent in every case, notwithstanding the amounts of the oxides of these metals in the furnace.

THERMAL BALANCES.

Effort has been made during the entire period of operations upon these alloys to construct a thermal balance upon the actual operating data. Accurate weighing, charging, sampling and chemical analyses were made. However, due to the lack of empirical constants for the specific heats of the elements involved, as well as the difficulty in determining the actual temperatures prevailing, nothing has been constructed that co-ordinates with actual results obtained.

At this point it might be stated that very little of the thermo-chemical and thermo-physical data that have been determined applies with any degree of accuracy. There appear to be certain undetermined factors in the mass action of the activities of electric furnaces not yet fixed to constants. For instance, it has been found that a charge calculated theoretically and giving expected results upon a 1,000 KV.A. furnace, produces entirely different results in a 3,000 KV.A. furnace. The lime in the charge of the smaller furnace is practically all recovered in the slag, whereas in the larger furnace the volatilization loss of lime amounts to 50 percent of that charged. Factors obtained from the data of operation of a 1,000 KV.A. furnace at 65 volts are not found applicable to the operation of a 3,000 KV.A. furnace at 90 volts, when treating identical raw materials for the same product.

USES AND PROBABLE FUTURE OF SILICON-MANGANESE.

Practically no authentic statistics are available upon the quantity, composition or specific uses of silicon-manganese in this country or abroad.

Harden states that 3,500 to 4,000 tons is produced in Sweden per year. A number of writers refer to the use of such alloys abroad, but give nothing tangible or specific.

Several hundred tons were produced by the Southern Manganese Corporation during the past two years. This, as stated above, was generally upon special orders, and as shown above covered a wide range of content of the various constituents. The control of the content of the various constituents specified was accomplished so readily that shipments were uniform within the limit of analytical error.

The most valuable and authentic paper that has come to the writer's attention, bearing upon this phase of the manganese silicon alloys, is that of Mr. Samuel L. Hoyt, entitled "Manganese Alloys in Open Hearth Practice," published in the March, 1919, issue of *"The Blast Furnace and Steel Plant."* Mr. Hoyt gives a resumé of data obtained from actual practice, covering a period of three years, and shows a parallel of comparative results of

the use of ferromanganese and ferrosilicon, as against that of silicon-manganese. Careful study of this article is recommended to those interested.

The conclusions and summary of Mr. Hoyt's observations are interpreted by the writer as follows:

(1) The average amount of manganese remaining in the steel, using ferromanganese and ferrosilicon, was 80.9 percent, while the average using silicon-manganese was 91.3 percent.

(2) "They show that the same results as to manganese and silicon content of the finished steel can be obtained by using silicon-manganese with smaller amounts of manganese and silicon, as compared to using the combination of ferromanganese and ferrosilicon."

(3) "There is the advantage of having a more uniform practice, which in itself would warrant smaller additions."

(4) "The low carbon content of the silicon-manganese may or may not be a material advantage, but it is in favor of the single alloy addition, because the carbon need not be worked as low, and there seems to be less danger of missing the carbon."

(5) "The indication of the results obtained in actual practice is that silicon protects the manganese in oxidized heats."

(6) "The manganese-silicon alloy would appear, in steel casting, to have the distinct advantage of making cleaner and sounder castings."

The last statement has the basis of scientific reasoning, since the portion of the alloy performing the function of deoxidizing, will, in the case of ferrosilicon, produce a certain amount of silica, a highly refractory oxide, which solidifies far above the temperature of molten steel, and has a natural tendency to remain in the ingot or casting as inclusions, which is very detrimental to the product. On the other hand, when the deoxidizer is applied in the form of a definite alloy, its oxidation product is that of silicate of manganese and possibly iron, with a melting point far below that of molten steel, and with a natural tendency to coalesce into larger aggregate and free itself from the steel.

The wide range of silicon-manganese alloys demanded by the trade shows that no definite lines of standard practice exist among

the steel workers. There will always be a certain degree of individuality in the metallurgical practice of steel metallurgists. If they adopt the use of silicon-manganese alloys as standard practice, they will no doubt demand a definite analysis of the alloy, and insist upon accuracy and uniformity in the quantity supplied. This can readily be furnished to individual specifications, as shown by the development of the industry during the past two years.

Some steel metallurgists of note have objected to the introduction of silicon-manganese into their works, due to the purely operating difficulty of having a third alloy, in their standard practice, which would possibly hazard its undesired charging at times by workmen. This objection is well founded in the light of the personal equation of the average melting force. On the other hand, should the metallurgists arrive at the standard silicon-manganese ratio desired, there could be furnished a standard alloy to specifications, which would bring the practice down to the use of one alloy instead of two.

The technical development of the American silicon-manganese industry has been established, and the product shown advantageous. The continuance of this and other ferro-alloy industries will depend largely upon the action of our national legislative body.

Anniston, Alabama,
March 29, 1919.

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ELECTRIC FURNACES OF THE RESISTANCE TYPE USED IN THE PRODUCTION OF ESSENTIAL WAR MATERIALS.

By T. F. Baily.[1]

Abstract.

A detailed description of a 900-K.W. heat-treating equipment intended for automatically and continuously hardening and tempering cast-steel anchor chains of the heaviest type. The heating is by an electrical resistor of granular carbon confined in carborundum fire-sand walls, and the machinery is controlled by the pyrometers, which allow of hardening and tempering at the exact temperatures to which the machine is set. At the meeting, the reading of the paper will be accompanied by motion pictures of the furnace of operation.—[J. W. R.]

The furnaces described in this paper are all of the resistance type, whose resistance element (resistor) consists essentially of carborundum fire-sand troughs, set on fire-brick piers within the furnace chamber proper, and filled with broken carbon, the electric current being led into the resistor at the end of each trough by means of the usual carbon or graphite electrodes.

The furnaces described may be divided into two groups: first, those for the annealing and heat-treating of steel, and second, for the melting of non-ferrous metals.

The first group includes furnaces built for the heat-treatment of cast-steel anchor chain for the Emergency Fleet Corporation, the heat-treating of crankshafts for Liberty airplane motors, the heat-treatment of small parts for airplanes, the heat-treatment of gears for motor trucks, and the annealing of cast-steel wheels for army trucks.

[1] President, The Electric Furnace Co., Alliance, Ohio.

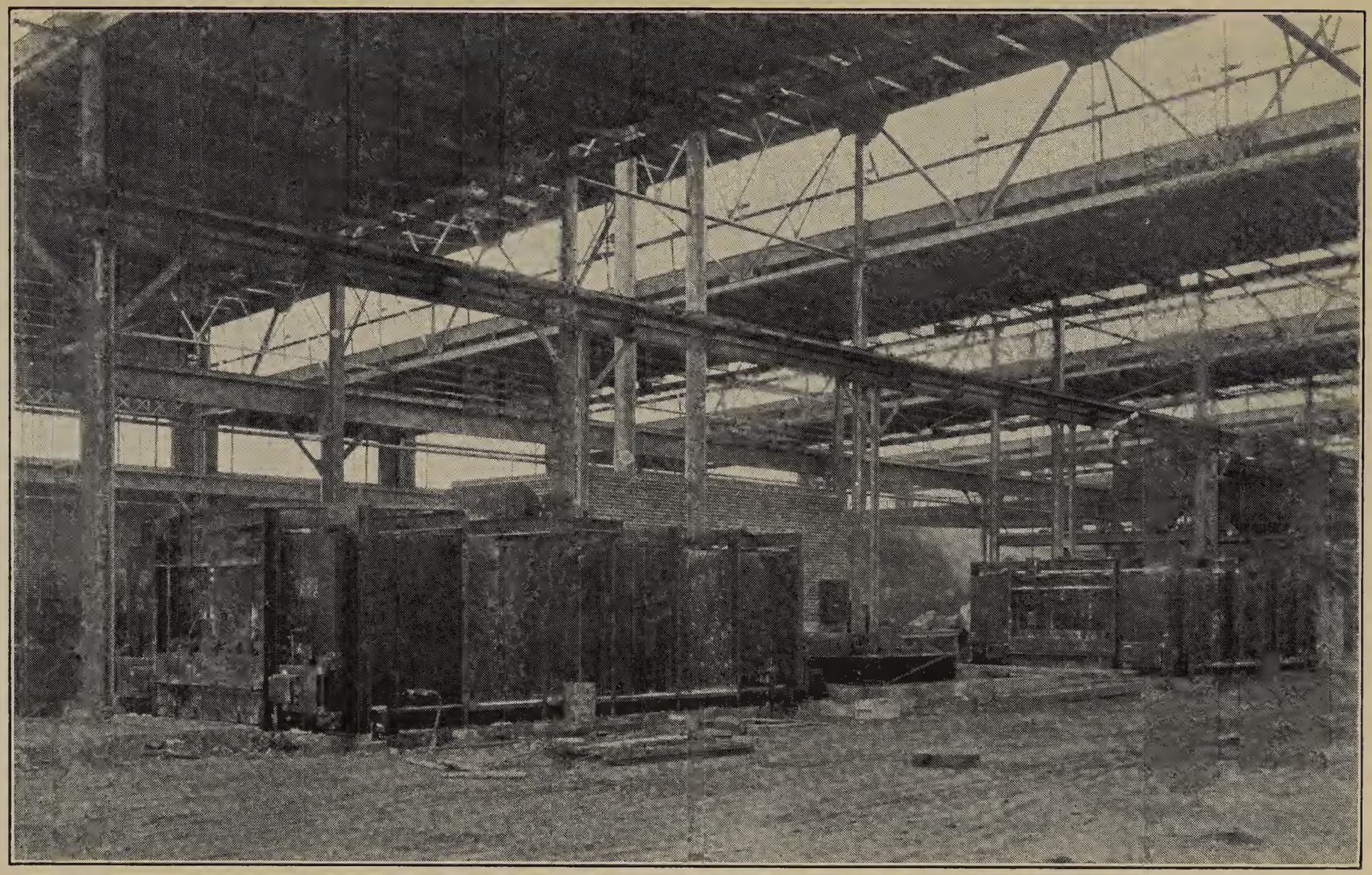
The second group embraces furnaces for making phosphor bronze castings for airplane motors, manganese bronze castings for submarine work, brass castings used in accessories entering into the contruction of cantonments, and metal to be used in extrusion billets for naval shells.

This paper, however, will deal specifically with the three 900-K.W. heat-treating sets supplied to the National Malleable Castings Company, Cleveland, Ohio, for the heat treatment of cast-steel anchor chain for the Emergency Fleet Corporation. The motion pictures to be shown later cover the first of these units installed, and the only one to go into service before the signing of the armistice.

Each of these equipments consists of one 600-K.W. furnace for heating the anchor chain for hardening, a 300-K.W. furnace of the same general dimensions for the drawing operation, and a concrete water-pit for quenching, as well as a motor-driven winch in front of each furnace for handling the chains into, through and out of that furnace.

The quenching pit is located between the discharge end of the first furnace and the entering end of the second furnace, and the three are directly in line, the movement of the chain proceeding directly from the hardening into the quenching bath, thence into the drawing furnace, and later onto the cooling floor at the end of the equipment.

The entire length of the equipment from the entering end of the hardening furnace to the winch at the discharge end of the drawing furnace is 132 ft. (40 m.). Each furnace is 28 ft. (8.5 m.) long by 16 ft. 9 in. (5.0 m.) wide. The cast-iron hearth for supporting the chains while being heated is 6 ft. 6 in. (2 m.) wide by 24 ft. (7.2 m.) long. The quenching pit is made of concrete and is 34 ft. (10 m.) long by 10 ft. (3 m.) wide and 4 ft. 4 in. (1.3 m.) deep. The chain, during the quenching, is supported on a grill work of 7 in. (18 cm.) inverted channels, which keep the chain submerged in 2 in. (5 cm.) of water. The quenching water is admitted through a group of jets supplied with water from the city water main, and located at the entering end of the quench pit, so that as the chain comes from the hardening furnace and enters into the quenching pit it receives initially a very rapid quenching at the time of preliminary immersion.



ELECTRICAL FURNACES FOR THE HEAT TREATMENT OF CAST STEEL CHAIN.
National Malleable Castings Co., Cleveland, Ohio.

Each of the winch mechanisms for handling the chains through the equipment consists of a motor-operated drum, similar to a crane trolley, and supplied with a 10 H.P. motor.

The transformer equipment for supplying electricity to the furnace consists of three 300-K.W. 60-cycle single-phase transformers, with their high-tension side wound for 11,000 volts. These transformers take their supply from the lines of The Cleveland Electric Illuminating Company. The secondary taps of these transformers are connected with the usual selective oil-break control switch of 2,000-ampere capacity, and thence to the furnace terminals of the transformers supplying current to the hardening furnace, one to each resistance trough, while the third transformer is connected in parallel to the two troughs of the drawing furnace.

These equipments are adapted to handle anchor chain from the 1-15/16 in. (5 cm.) section chain used extensively by the Emergency Fleet Corporation, up to and including the 4½ in. (11.3 cm.) battleship chain. The production, however, so far has been almost entirely on the standard chain for the Emergency Fleet.

As the chains are all of 90 ft. (27 m.) length, and must therefore be treated in sections of this size, each chain is folded into four lengths substantially 22 ft. (6.6 m.) each, and two of these 90 ft. (27 m.) lengths are handled through the furnace at a time.

In order to reduce the labor to a minimum, and prevent the placing of cold manipulators on the hot chain for quenching, four cast-steel hooks are fastened to a steel cross-bar member, and the folds of the chain are looped over these cast-steel hooks before the chain is drawn into the hardening furnace.

In starting the operation a long steel hook is run from the entering end of the hardening furnace entirely through that furnace, where the pulling chains from the first winch located on the quenching pit are hooked onto it. The pulling chains of the winch are backed off the drum and pulled entirely through the furnace at which time they are hooked onto the cross-bar of the cast-steel hooks just mentioned. On signal to the winch operator, the winch is reversed and the two 90 ft. (27 m.) lengths of chain, weighing approximately 3,400 lb. (1,550 kg.) each, are drawn into the furnace and the door closed. Four slots are cut in the door of the discharge end of each furnace, to provide space for the cast-steel hooks when the door is down, it being understood

that about half of the cast-steel hook remains in the furnace and becomes heated to the temperature of the chain, while the cross-bar comes clear of the furnace and remains in the open air.

The chain is allowed to remain in the hardening furnace for approximately 2 hours, in which time it has come fully to temperature, which is approximately 850° C., at which time the doors are opened and the winch with its pulling chains, which are still connected to the cross-bar fastened to the pulling hooks, pull the heated chain forward, where it is rapidly immersed in the quenching bath, the quenching water being introduced at the end of the quenching pit where the chain enters and consequently where most of the heat must be dissipated. The heated water passes over a spillway at the outside end of the pit. Before the chain is fully cooled, the pulling chains from the winch of the drawing furnace are pulled through that furnace in the manner described in the similar operation on the hardening furnace, and hooked onto the cross member of the main connecting hooks holding the anchor chain. The hooks from the first winch are at this point disconnected and the quenched chain drawn into the drawing furnace where it remains for a similar period of time to that in the hardening furnace, after which it is drawn by the winch onto the cooling floor beyond, where, subsequent to its cooling, it is taken to the testing machine.

The heat treatment of this anchor chain is essentially the same as that used by the same company in the treatment of their draw-bar knuckles for railway couplers, the steel being of the same analysis, and the equipment just described is a special adaptation of the two somewhat analogous electric heat-treating equipments supplied to this company's Sharon plant.

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